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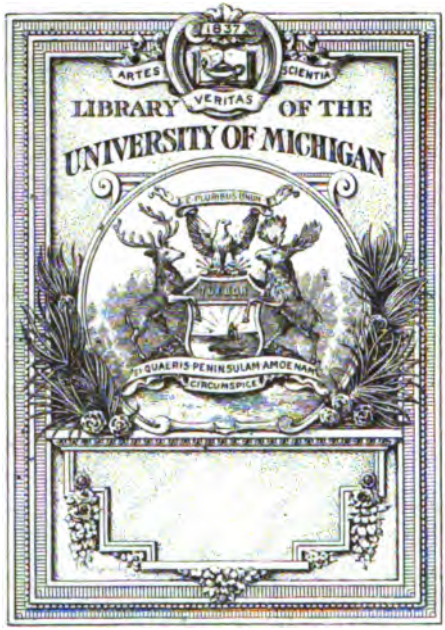

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THE

SCHOOL OF MINES

QUARTERLY.

VOL. III.

NOVEMBER, 1881, TO MAY, 1882.

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THE

SCHOOL OF MINES QUARTERLY.

VOL. III. NOVEMBER, 1881. No. 1.

MEMORANDUM AS TO THE PROPOSED COURSE OF ARCHITECTURE IN THE SCHOOL OF MINES.[1]

BY THE PROFESSOR-ELECT OF ARCHITECTURE.

THE chief difficulty in the study, as well as in the practice of architecture, is its many-sidedness. There is hardly anything which, in theory, an architect should not know. In the practice of the profession, of course, every man simplifies the problem for himself, by devoting himself mainly to one side of the subject, one distinguishing himself chiefly in design, another in practical affairs, a third in the applications of science to the art. But a school cannot so narrow its range; and although, in fact, the French courses of study are mainly artistic, and the German scientific, and the English practical, they all, from this very fact, fail to furnish the model we should wish to follow.

The problem before us in this country is to devise a course of study so carefully adjusted that the practical, scientific, and artistic studies may receive equal consideration. Ultimately, after he gets out of school, the student will prosecute his studies in one way or in another, as the bias of his mind may determine, and he will have plenty of time for this. The period

[1] From the President's Annual Report. See also abstract of this report in the School of Mines Notes.

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of pupilage lasts, for the architect, until he is thirty years old; for, though he is able to support himself before that time, he is not till then, in general, ready to take an independent position. Up to that time he is in the attitude of a student. If this is the case already with young men whose training is accidental and imperfect, it will certainly be true of the thoroughly educated men which it is the object of a first-rate school to produce. All the time they can gain, through better adjusted means of study, will be needed for the exploration of the wider field opened to them. The time spent at school is then not more than a third or a half of their term of professional preparation; and it should be spent, it seems to me, first in learning good methods of study and of work, and then in the acquisition of a minimum of knowledge and skill in each of the three departments I have indicated, not neglecting, at the same time, the languages and literature. But these studies should, I think, for professional students, be directed along special professional lines, and made to co-operate with the critical and historical work of the professional course.

A considerable restriction of the range of study would, of course, be imposed by the number of branches to be taken up. The student of architecture cannot, for instance, follow the full course of scientific construction taken by the student of civil engineering, and have any time left for the artistic and practical studies which are, after all, the distinctive characteristics of his work. But it is also necessary to limit the range of school work, in order that what is learned may be made so familiar that it can never be forgotten. The main principles of scientific construction, for instance, and their applications in practice, should be so reiterated and enforced as to seem matters of course, so that they may not lie in the mind as something external to it, likely soon to drop away, but may be thoroughly assimilated, instinctively controlling the judgment and informing the common sense. Then the student whose later studies lie wholly in the field of design will spontaneously conform to the requirements of construction, and the student who gives himself mainly to practical affairs will comprehend and adapt himself to the requirements of design. Later in life two such men can comprehend each other's work, as can never happen where the mere engineer and the mere designer try to co-operate with each other.

How the artistic and historical studies of such a course had

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best be conducted is not a very difficult question. The experiments we have here been trying, partly founded upon the example of the School of Fine Arts in Paris, very well indicate, both in their successes and through their failures, how the elements of architecture, as an art of design, may be systematically taught, and with what illustration of theory and of historical examples they may conveniently be accompanied.

The scientific side of the subject is not so easily disposed of. The principles of scientific construction are, of course, the same for the architect and for the engineer. But the student of architecture is at a disadvantage, compared with the student of engineering, both in understanding them and in applying them. All the text-books are written from the point of view of the engineer, all the examples relate to engineering work. When the engineer gets through his studies and takes up the practice of his profession, he finds himself quite at home, his mathematics and mechanics having made him quite familiar with the field in advance. The applications of science to his art are already familiar to him. The student of architecture, on the other hand, finds a great gulf fixed between the region of scientific studies and the world of his professional life. His statics and dynamics seem unrelated to his daily problems. The subject-matter is the same, but the point of view is different, and, unless he is distinctly of a scientific and engineering turn of mind—which he had better not be, in most cases—the task of establishing a connection between the two will be beyond his powers.

What is needed in a course of architectural education is that the scientific studies necessary to an architect shall be pursued throughout from an architectural point of view; that the chemistry, physics, and mechanics should at every turn have their bearing upon the arts concerned in building clearly pointed out. Whether this can best be done by simply furnishing special examples and applications, by way of notes to the ordinary text-books, or by more specialized courses of scientific study—this is one of the questions which is waiting to be answered. If the answer were obvious, experiment and investigation would not be needed.

The third point to be made in a satisfactory scheme of architectural study, though more novel, perhaps, presents fewer difficulties. This is the study of practical matters, of the

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arts ancillary to architecture, the arts of the plumber, painter, mason, plasterer, etc. It is not necessary that an architect should be a skilful workman at these trades; but he does need to understand them technically, and to be a good judge of work in each. At present it is difficult for a student to learn, in any way that is possible for him to remember them, even the elements of such knowledge. Office life, of necessity, gives him but little opportunity to accomplish himself in these respects. The only way to secure this seems to me to be by means of a properly organized laboratory, in which the principles of these arts can be learned by handling the tools; nor can I doubt that what is thus learned will be easily remembered. Such a workshop need not be expensive either in its equipment or in its administration, and it would do more than anything I can think of to strengthen the profession in what is now its weakest point—and it is to improve the profession that schools of architecture are founded.

The question is sometimes raised whether, architecture being counted among the fine arts, it does not belong in a school of art rather than in a school of science. But if a thorough and comprehensive course of study is to be established, a school of science seems the most convenient place for it. Two out of its three branches are certainly more germane to scientific pursuits than to painting and sculpture, and it is easier and cheaper to add the apparatus needed for the study of elementary design to a school of science than to bring the workrooms and laboratories of a school of science into a school of art. Still, it needs to be distinctly recognized that the atmosphere of exact science is unfavorable to the growth of the artistic sentiment; and that in temper and methods a school of architecture must always be, so far as relates to design, at least, not quite at one with the purely practical schools with which it is associated. It must accordingly require special pains to create for it an atmosphere of its own, favorable to the harmonious development of its own students.

WILLIAM R. WARE.

BOSTON, March 5, 1881.

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THE GENESIS AND DISTRIBUTION OF GOLD.

By Prof. J. S. NEWBERRY.

GOLD occurs in three classes of deposits: 1st, placers; 2d, segregated quartz veins in metamorphic rocks; 3d, in fissure veins, or repositories connected with them, where it is mingled with the ores of iron, lead, tellurium, silver, copper, etc.

The origin and mode of deposition of the gold in these deposits have been discussed at considerable length by geologists, chemists, and mineralogists, with such a lack of harmony on some important points as to leave the student in uncertainty what to believe. Hence, I have thought that the observations and convictions of one who has had an opportunity of studying most of the important gold deposits of our country, and who is fresh from the field, might not be unwelcome to the QUARTERLY. I shall therefore give, on the following pages, a brief review of the subject as it appears from my standpoint.

PLACER DEPOSITS.

Nine-tenths, and possibly ninety-nine hundredths of all the gold which has come into the possession of man, has been obtained from superficial deposits. These are called *placers* by the Spaniards, from whom we have adopted this as we have so many other mining terms.

The gold here occurs in beds of sand, gravel, and boulders which have plainly been washed down from higher ground, usually from neighboring mountain slopes composed of metamorphic rocks containing auriferous quartz veins. This relationship is so constant that most persons who have observed or read much of placer gold deposits have been satisfied with the simple and obvious inference that the gold has been derived from the erosion of higher rocks and the breaking down of quartz veins, the detrital materials having been sorted by water according to their specific gravity. Probably this view would have been universally accepted, the array of facts in its support is so convincing, if it had not happened that men are so constituted that it is impossible for all to think alike on any

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subject. Unfortunately, there are in all communities intelligent men who have a proclivity (we are, perhaps, justified in calling it a perversity) that leads them to choose instinctively the opposite of the generally accepted view, or to reject a simple and easy solution of a problem in favor of an intricate and difficult one. In this phase of human nature we have another illustration of what most persons have experienced: that the artificial difficulties placed by egotism or unreason in the way of the searchers for truth are more difficult to surmount and more trying to patience than those which nature has opposed. Among other fruits of this constitutional heterodoxy is the coinage of a new theory to account for the genesis of gold in placers, viz., that the grains and nuggets are formed where found by precipitation from chemical solution.

The facts by which the advocates of this theory have attempted to sustain it are: 1st, the rarity of nuggets or gold masses of any considerable size in quartz veins; 2d, the greater purity of the fine gold of placers than that of the neighboring veins; 3d, the frosted character of the surface of some gold nuggets; 4th, instances of deposition of gold in organic substances buried in the gold placers; and 5th, the solubility of gold as proved by laboratory experiments.

In arguing from these facts, it is stated that nuggets of gold of large size have been not unfrequently met with in all the great placers: such as the "Sarah Sands" and "Welcome" nuggets of Australia, which weighed respectively 233 and 184 pounds; the great nugget from Miask, in the Ural Mountains, which weighs 96 pounds; and others ranging from twenty-five to fifty pounds, taken from the placers of Australia, New Zealand, and California; while "no masses of gold of anything like the size of these have been found in quartz veins." This latter statement is, however, erroneous, for the largest mass of gold yet found on this continent was taken from a quartz vein in the Monumental Mine, at the Sierra Buttes, twelve miles north of Downeyville, Cal. It weighed 95½ pounds, and is said to have been originally larger, weighing 140 pounds. This discovery is well authenticated, and is reported in detail in Prof. Raymond's Report on Mines and Mining for 1870 (Ex. Doc. No. 207, page 63). This, then, was one of the largest nuggets of gold found in the world, and proves that large nuggets can and do occur in quartz veins. Nor is this rarity of nuggets in quartz veins any proof of the truth of this

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new theory, for it is certain that the amount of nugget gold taken from mineral veins is in fair proportion to the whole gold yield of such veins when compared with the proportions of nugget and dust gold in the placers. When it is remembered that from the slopes of the Sierra Nevada, over large areas containing auriferous quartz veins, a sheet of material, perhaps thousands of feet in thickness, has been removed by erosion, it will not be surprising that a considerable number of masses of gold have been taken from the resulting *débris;* and when we compare the insignificant excavations made by man in quartz veins with the gigantic mining operations of nature, we can only wonder that he has met with any of these large and rare masses of gold. The nugget argument, therefore, has no support, and falls to the ground.

It is true, as has been asserted by the advocates of the chemical theory of the genesis of gold, that the fine gold in some placers is of higher grade than that of the neighboring quartz veins. This difference is, however, not marked, and something of this sort was inevitable, from the fact that all gold in quartz veins and placers is alloyed with silver. When minutely divided and exposed to the action of chlorine and other agents which dissolve silver—so generally prevalent in the atmosphere and superficial deposits that silver is never found there in the metallic state—some portion of the silver would naturally be removed from the surface of the particles of gold.

It should also be said that the *fine* gold of placer deposits is usually purer than the coarse gold and nuggets, and the reason is that the scales and fine grains of gold present more surface to be acted on by the agents which remove the silver. This process, which is essentially that called *pickling* by the jewelers, with the fact that gold itself does not *perfectly* resist chemical agents, also affords an explanation of the frosted appearance which the surface of some placer gold exhibits, and which has been hastily taken as proof that all such masses were chemically precipitated.

Gold is usually said to be soluble only in aqua regia, and the idea is prevalent that it would remain forever unaffected by any of the chemical agents in nature's laboratory. This is, however, far from true, as by the experiments of modern chemists it has been shown that it forms a great variety of chemical combinations. J. P. Pratt, epitomizing the results of his experiments, says (*Jour. de Pharmacie et de Chimie*,

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August, 1870): "Gold can be readily oxidized and salified by oxacids. There exists a liquid and volatile chloride of gold containing more chlorine than the sesquichloride. There exists, likewise, a sesquioxide and a carbonate of gold; and, lastly, gold behaves in many instances like some of the other metals." While all this is undoubtedly true, and it is also true that in the gossans of some of our Western mines gold exists in other than metallic form, these facts lend no confirmation to the theory of the chemical origin of placer nuggets. A more interesting fact, proving the solubility of gold by natural processes, is reported from the placers of California, viz., that the bark of some of the tree-trunks found buried in the blue gravel was largely replaced by iron pyrites, which was rich in gold. Hence, we cannot deny that some gold has been deposited in the placers from solution; but this certainly does not include the nuggets and gold-dust. The facts which distinctly militate against that theory are briefly as follows:

First.—Deposits of placer gold are always found adjacent to and lying below districts traversed by auriferous veins, and nowhere else.

Second.—The areas where the quartz veins occur have certainly suffered great erosion, and mechanical forces have there been in action tending to break down and comminute the quartz, and to liberate and wash the contained gold.

Third.—The conditions in which the placer gold is found—viz., mingled with rolled fragments of quartz and in the irregularities of the surface of the bed-rock, where a washing process on a large scale has been in action, and where such washing processes would have left it—prove the accumulations of gold to be mechanical rather than chemical. A deposit from chemical solution would not thus be circumstanced and localized.

Fourth.—The distribution of gold in placer deposits is also demonstrative of its mechanical origin, for in all cases known to me the nuggets and coarsest gold are found nearest the outcrops of the quartz veins that have supplied them; while the particles become gradually finer and finer as the line of drainage is followed from this point. Hundreds of instances of this kind might be cited—enough, indeed, to form the basis of a

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mechanical law in itself an unanswerable argument to the chemical theory.

Fifth.—Nothing is more common than to find, in the placers, pebbles and fragments of gold-bearing quartz, which must have been derived from the neighboring veins, and most of the nuggets have more or less quartz, just like that of the veins, still adhering to them.

Sixth.—The surfaces of nuggets almost always bear incontestable evidence to the battering they have sustained. They are generally rolled and rounded, and the surface is such as could be produced only by blows and friction.

The cases where the surface of the nuggets is rough and frosted, as though from partial solution, are extremely rare, and afford no support to the chemical theory.

Seventh.—If the gold of placers were deposited from solution, we should necessarily find much of it crystallized and forming strings and sheets running through the porous material; whereas, as a matter of fact, crystals are never found in placer gold, nor are sheets or thread. Scales, grains, pebble-like nodules, rounded, battered masses, these are what we find; in other words, sand, gravel, and boulders of gold formed and transported by mechanical means.

I cannot better illustrate the occurrence of gold in placers than by describing an isolated placer I have recently visited, which combines all the normal features of this class of deposits, and bears, in unmistakable characters, the record of its history. It is located at Osceola, Nev., and lies on the west flank of Mount Wheeler, said to be the highest mountain in the State. The central mass of Mount Wheeler is granite, but on its flanks occur upturned and metamorphosed paleozoic rocks, quartzites, slates, and limestones. A spur, which extends north from the main peak, consists at its northern extremity of high, rough, and ragged masses of quartzite. These are succeeded on the south by a belt of talcose slate several miles in length, forming a ridge which rises four thousand feet above Spring Valley, its western boundary. At its southern extremity the slate belt is overlain by limestone. All these rocks are cut by veins of quartz, but those in the quartzite and limestone seem to be barren. The quartz veins in the slate belt are, however, numerous, and many of them are rich in gold.

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In the course of ages the slate area has been extensively worn away, forming a "cirque," or semicircle, drained by several gulches, which combine below in one, and through which all the material removed from the mountain side has been spread over the slope to the valley below. In some places the mass of *débris* is three hundred feet or more in thickness; above it is narrowed between ridges of rim rock, below spread out into a fan-shaped delta. The detrital material consists chiefly of rolled boulders and pebbles of quartzite and vein quartz mixed with sand and a cementing clay derived from the decomposition of the slate. All this contains gold: near the head of the gulch that which is coarse; that below becoming finer and finer toward the valley.

The climate of this region is now excessively dry, no water flowing through the gulch, except during a week or two in the spring, when the snows are melting on the mountain above. The temporary streams thus formed have cut narrow channels through the beds of gravel and boulders, in the upper part of the gulch to the bed-rock, below only in the loose material. These modern channels have revealed the existence of others that are much older and broader, now filled, and in places deeply buried. These old channels would seem to have been produced at a time when the climate was more moist and the flow of water from the mountain much greater than now. As usual in such cases, the old channels are rich placer ground, the gold having accumulated in the depressions of the irregular bed-rock.

The number of nuggets and the quantity of coarse gold taken from this placer is remarkable. The largest nugget found weighed twenty-four pounds, and many others have been met with weighing from half a pound to two pounds. At the time of my visit two nuggets, weighing over ten ounces each, were purchased which had been found by one man on September 15th. Between two and three hundred thousand dollars have been taken from this placer within the last four years, all by hand-labor, and for the most part during the brief interval when a little water was flowing down the gulch. The nuggets have all been found toward the head of the gulch. Many of them have vein quartz still adhering to them, and their derivation from the quartz veins which crop out on the mountain side above cannot be doubted. The whole of this placer has been purchased by parties who are about to bring water on to it

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from streams which drain the southern slope of Mount Wheeler. These streams run in gulches, which have the same general character with that at Osceola; but none of them contain gold, simply because the slopes they drain are not composed of auriferous rock.

The significance of the facts I have given, and their bearing on the question discussed on the preceding pages, may be briefly summed up as follows: Mount Wheeler is a very high mountain, which, though located in an exceedingly dry region, has through ages received on its summit sufficient precipitated moisture to form torrents that drain and have deeply scored its different sides. The material excavated is all spread in the vicinity, forming slopes of gravel and boulders at the mouths of all the gulches.

In one of these gulches the detritus is rich in gold; in all the others it is barren. The rich one has received all the *débris* from a portion of the mountain, composed of talcose slate cut in every direction by auriferous quartz veins. The other gulches contain no gold, and there are no gold veins at their heads. In the rich gulch the gold becomes coarser and coarser toward the head as the outcrops of the quartz veins are approached. When to this is added that not a crystal of gold has been found in this placer, nor any gold that could be fairly considered a chemical deposit—but, on the contrary, all the masses are rolled and battered, often with adhering quartz—we have an array of evidence in favor of the mechanical deposition of the gold which cannot be gainsaid.

GOLD IN SEGREGATED VEINS.

Most of the quartz veins which carry gold belong to the class of what are called segregated veins. These occur only in metamorphic rocks, are lenticular sheets, limited in depth and lateral extension, and generally showing little of the banded structure so characteristic of fissure veins. They consist mainly of quartz, in which the gold is sometimes free, but more commonly contained in iron pyrites, with which yellow copper is often associated. Sometimes the gold is not strictly confined to the quartz veins, but extends more or less into the enclosing rocks, which are oftener than otherwise magnesian slates.

The gold in segregated veins would seem to be indigenous

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to the formation in which it occurs, and not, as in fissure veins, to have been derived from some foreign source. It is usually supposed that, before they were metamorphosed, the rocks which enclose the segregated veins contained gold generally, though sparsely, disseminated through them, and that, in the process of the segregation of the siliceous matter to form sheets of quartz, the gold was somehow gathered and concentrated by it.

Sir Roderick Murchison, guided by his study of the gold deposits of the Ural Mountains, supposed that auriferous quartz veins were confined to paleozoic rocks, but that the gold impregnation had taken place at a comparatively recent date. It was demonstrated, however, by Prof. Whitney, in the prosecution of the geological survey of California, that the metamorphic slates which carry gold in the Sierra Nevada are of Triassic and Jurassic age; and in the light of later observations we may say that metamorphic rocks of all ages contain auriferous veins. Nearly all the great mountain chains of the world contain more or less of such veins, and as these mountain chains have been the great condensers of moisture, and erosion has been constantly wearing down their slopes, placer deposits have been formed which have supplied most of the gold yielded by the earth to man. As it can be procured from them by the simplest methods, the work of its extraction was begun by prehistoric races, and the Altai, the Himalayas, the Ural Mountains, the Australian Alps, the Sierra Nevada, and the Rocky Mountains, have in turn contributed their millions to the treasuries of the world. These mountain chains are of very different ages, and we have abundant evidence that gold has existed in some of them from the earliest geological times. The oldest mountains of which we have any knowledge—the Laurentian, of Canada, now nearly removed by erosion—contained auriferous quartz veins that have supplied gold to all the successive formations which have been derived from their ruins. The gold impregnation of the Laurentian rocks dates back certainly to the period of their metamorphism; and this was pre-Silurian, for the undisturbed lower Silurian strata overlap and partially cover these gold-bearing rocks.

In the same way the gold at the Black Hills is proved to be pre-Silurian, since the Potsdam sandstone which abuts against the Archean nucleus of the hills in places contains rolled fragments of the Archean rocks, and gold washed from them in such abundance as to form rich mining ground—the

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so-called cement deposits of that region. The distribution of gold from the Archean rocks has probably been constantly going on from the Silurian age to the present day. This is shown in the almost universal dissemination of gold through the drift of New England, New York, Ohio, etc., where the superficial materials have been largely derived from the Canadian highlands. In Ohio gold is found in the drift clays, sands, and gravels, and locally in as great quantity as in the poorer placers of California. There is little doubt that the mechanical sediments derived from the wear of the Archean rocks all contain gold, and since it has been proved that gold exists in sea-water, it has probably impregnated all the organic marine sedimentary rocks as well. In the subsequent metamorphism of some of these strata it has been concentrated in such a way as to produce auriferous quartz veins rich enough to be worked.

From these facts it will be seen that there is no geological age which can be called the age of gold. It existed in the oldest rocks known, and from them and their derivatives, more modern rocks, it has been and is now being constantly distributed by both mechanical and chemical processes. Even some of the igneous rocks of the Western country are said to contain minute quantities of gold;[1] and this is not surprising, if, as is supposed, much of our volcanic material is a fused condition of sedimentary rocks.

GOLD IN FISSURE VEINS.

As is well known, gold is a frequent constituent of the fissure veins of the Far West. The ore of the Comstock vein has yielded about 47 per cent. of gold and 53 per cent. of silver; and it is probable that one-half of the so-called *silver veins* contain gold in sufficient quantity to be of practical value. In some true fissure veins gold is the only valuable ingredient, but more generally it is associated with several other metals. The Revenue Mine, at Tuscarora, Nev., contains silver in the form of arsenical and antimonial sulphide, and gold in iron pyrites frequently crystallized lining cavities. At Eureka the ore occurs in chambers, which were originally filled from a

[1] *E.g.*, the basalt of the Snake River lava-plain, and, according to Prof. J. J. Stevenson, the trachyte of Colorado.

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solution issuing through fissures from below and deposited as argentiferous galena and auriferous pyrites, the silver and gold being in nearly equal proportions. In the great veins of Bingham Cañon, and at the Cave Mine, near Frisco, in Utah, the combination is the same, and, as at Eureka, the sulphides have been decomposed to a spongy, rusty gossan. At the Bassick Mine, in Colorado, gold exists free, or in combination with tellurium and associated with zinc, copper, and iron. In all these, and many other cases which might be cited, the gold has been brought up in a hot solution impregnated with mineral matter far below, and deposited as the temperature and pressure were reduced. The formation of this class of auriferous deposit is well illustrated by the Steamboat Spring, in Western Nevada, where hot water, flowing out through fissures produced by subterranean forces, is depositing a siliceous vein-stone, containing sulphides of iron, copper, oxide of manganese, and metallic gold. There is little doubt that, in the great mineral belt lying between the Sierra Nevada and the Rocky Mountains, where, in Tertiary times, volcanic activity was exhibited on a grand scale—sedimentary rocks upheaved and fissured in every direction, with great outflows of fused material—hot springs, like the Steamboat, were everywhere busy, doing similar work. Bursting out at different places and times, and flowing from different sources, the solutions they carried and the ores they deposited varied greatly; but the methods of accumulation, transportation, and deposition were essentially the same, namely, the leaching of various rocks by steam and hot water under great pressure, by which silica and sparsely disseminated metals were gathered and driven toward the surface to be deposited as the pressure and temperature were reduced. Gold collected in this manner was unquestionably taken into chemical solution, and in the resulting vein deposits we find it in strings, scales, and irregular masses, often beautifully crystallized and associated with other crystallized minerals which are certainly chemical precipitates.

We may sum up the teachings of geology in regard to the genesis and distribution of gold by saying:—

First.—Gold exists in the oldest known rocks, and has been thence distributed through all strata derived from them.

Second.—In the metamorphosis of these derived rocks it

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has been concentrated into segregated quartz veins by some process not yet understood.

Third.—It is a constituent of fissure veins of all geological ages where it has been deposited from hot chemical solutions which have leached deeply buried rocks of various kinds, gathering from them gold with other metallic minerals.

Fourth.—By the erosion of strata containing auriferous veins, segregated or fissure, gold has been accumulated by mechanical agents in placer deposits, economically the most important of all the sources of gold.

[The following notes, from reports made by Prof. H. S. Munroe to the Japanese Government, embodying the conclusions formed after two years careful study of the gold placers of Yesso, may be of interest in this connection.

Gold-bearing gravel is found on the Island of Yesso in many of the river valleys, apparently derived in every case from the metamorphic strata of the immediate vicinity. The gravel, when auriferous, is always composed of pebbles of metamorphic rock, similar in composition to the metamorphic rocks of the upper valley of the stream. Wherever in the gold regions the river valley is mainly or entirely within the limit of the soft strata, the gravel is composed chiefly of fragments of sandstone and shale, and contains little or no gold. An interesting case, demonstrating these points, was developed in the survey of the Toshibets Gold Field. The Toshibets is formed by the junction of three large streams, two of which head in a range of metamorphic hills, while the third has its valley entirely within an area of Tertiary clay rocks and tufas. Tests of the gravel brought down by these three streams, the samples being taken in or near the main valley, gave the following results:

	Per cent.	Per cent.	Per cent.
Quartz pebbles in the gravel	34	6	12
Quartzyte, granite, gneiss, mica, and chlorite schist	66	85	11
Volcanic rock	none	3	none
Clay rocks and sandstone tufas	none	6	77
	100	100	100
Gold obtained per cubic yard	6⅛ cents	3⅛ cents	$\frac{1}{100}$ cent

These auriferous gravels everywhere afford unmistakable evidence of having been deposited in running water; and the direction of the old current, which can usually be determined from the position of flat stones in the beds, coincides, as a rule, with the general course of the present river valleys. The gold is found to be concentrated in the lower layers of the gravel beds and next the bed-rock. This concentration is undoubtedly due to the repeated stirring and rewashing of the gravel as the old river shifted its bed from one side of the valley to the other. The richness of the gravel and the coarseness of the grains of gold also increase very perceptibly as we ascend the stream—indicating unmistakably the source of the gold.—ED.]

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AN APPARATUS FOR THE RAPID ANALYSIS OF MIXTURES OF GASES.

ARTHUR H. ELLIOTT ('81), PH.B., F.C.S.

IN gas-works, and also in many iron- and steel-works, it often becomes necessary to make a number of analyses of mixtures of gases daily, which, of course, is only possible with simple apparatus. The earliest and simplest form of apparatus was a tube for the determination of carbonic acid, very much like an inverted burette, used by C. Stammer.[1] This tube was further modified by F. M. Raoult,[2] who used two stopcocks, one above and another below; and to the upper stopcock was attached a funnel, which serves to introduce the chemicals in solution for treating the gas. After treating the gas, Raoult washed out the chemical used with water, introduced through the funnel, and the gas was measured by bringing the tube to a nearly horizontal position and allowing water to run in through the funnel.

Wilkinson modified this tube of Raoult by placing it in a wider tube having a stopcock below, and omitting the stopcock upon the lower end of the measuring tube. By this means he could adjust the pressure upon the gas by adding water to or drawing it from the exterior tube.

But the difficulty with these methods is the necessity of washing out the chemicals used to absorb the various gases. For example, after treating a gas mixture with pyrogallate of potassium, it is necessary to wash out all the pyrogallate before adding bromine to absorb the illuminants, otherwise the bromine is absorbed by the alkali before it can act upon the gas. To obviate this difficulty, and one or two others, I have used the apparatus described further on.

The apparatus of Orsat at once suggests itself as a remedy for these difficulties; but, unfortunately, to attain the same accuracy as with the apparatus I describe, you have to use four times the amount of time, and moreover the apparatus of Orsat is both costly and fragile.

The apparatus is shown in the drawing on page 19.

[1] Dingler's Polytech. Jour., CII., 368. [2] Comp. Rend., 1876, 844.

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The tube A is of about 125 cc. capacity, whilst B, although of the same length, holds only 100 cc. from the mark D, or zero, to the mark on the capillary tube at C, and is carefully graduated into $\frac{1}{10}$ cc. The attachments to these tubes below are seen from the drawing, except that the stopcock I is three-way, with a delivery through its stem. The bottles K and L hold about one pint each. The tubes A and B are connected above with one another, and with the funnel M by capillary tubing, about one millimetre in internal diameter. There is a stopcock at G and another at F, while the funnel M, holding about 60 cc., is ground to fit over the end of F above. At E is a piece of rubber tubing uniting the ends of the capillary tubes, which are ground off square to make them fit as close as possible.

In beginning the analysis of a mixture of gases, the stem exit of the three-way cock I is closed by turning it so that L and A are connected through the rubber tubing; the stopcocks F and G are opened, and water is allowed to fill the apparatus from the bottles K and L, which have been previously supplied. When the water rises in the funnel M, and all air-bubbles have been forced out of the tubes, the stopcocks F and G are closed, the funnel M removed, and the tube delivering the gas to be tested attached in its place. By now lowering the bottle L slowly, and simultaneously opening the stopcock F, the tube A is nearly filled with gas, and the stopcock F is closed. The tube delivering the gas is now removed, the funnel M replaced, the bottle L raised, the bottle K lowered, and by opening the stopcock G the gas is transferred to the graduated tube B. By placing the bottle L on a stand at about the level of the water in A, the level in B and in the bottle K can be adjusted to the zero point, and the stopcock G is closed. The excess of gas in A is expelled by opening the stopcock F and raising the bottle L. The gas remaining in the capillary tube between C and the vertical part is disregarded, or in very careful work it may be measured and an allowance made in not filling the tube B quite to the zero mark, but usually it is too small to be worth notice.

Having measured the gas to be tested, it is now transferred by means of the bottles K and L into the tube A, and the fluid chemicals added by placing them in the funnel M and allowing them to flow down the sides of the tube slowly, care being taken *never* to let the fluids run below the level of the

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top of the vertical tube in the funnel. It is best to have a mark on the outside of the funnel at least three-fourths of an inch above the top of the level of the vertical tube, and never to draw the fluid down below this point.

Having treated the gas with the chemical, it is transferred by means of the bottles to the tube B, to be measured. If the chemical gets into the horizontal capillary tube, the passage of a little water from the bottle K will remove it, before transferring the gas. When the gas residue is in B, and the fluid in A has been adjusted at the mark C on the horizontal tube, the stopcock G is closed, the bottle K is lowered till the level of water in it and that in the tube B are the same, and the reading is then made. The tube A is now filled with the chemical just used and water. By turning the stem of the three-way cock I, so that it communicates with A, and also opening the stopcock F, the contents of the tube can be run out, and water run through the funnel M to clean the tube for a new absorption. When the tube is clean, by turning the stopcock I, so that A and L communicate, the water is forced into A, and the whole is ready to receive the gas for new treatment. By this means the gas is removed from the action of the water used to wash out the chemicals, and the chemicals are completely removed from any interference with each other when treating a mixture of gases. In using this apparatus the solutions are added in the following order:

1. *Potassic hydrate* to absorb carbonic dioxide (also hydrogen sulphide and sulphurous oxide[1] if present).

2. *Potassium pyrogallate*, to absorb oxygen.

3. *Bromine*, to absorb illuminants, like olefiant gas and acetylene, and after the absorption is complete, and the bromine vapors cause an expansion, a little potassium hydrate is added, to absorb these vapors before the gas is transferred and measured.

4. *Cuprous chloride* in concentrated hydrochloric acid solution, to absorb carbonic oxide. After this absorption is complete, the gas is transferred to the measuring tube, the contents of the tube A run out, the tube washed and filled with water from the bottle L. The gas is now transferred to

[1] If these gases are present in large quantities, special methods are necessary for their estimation.

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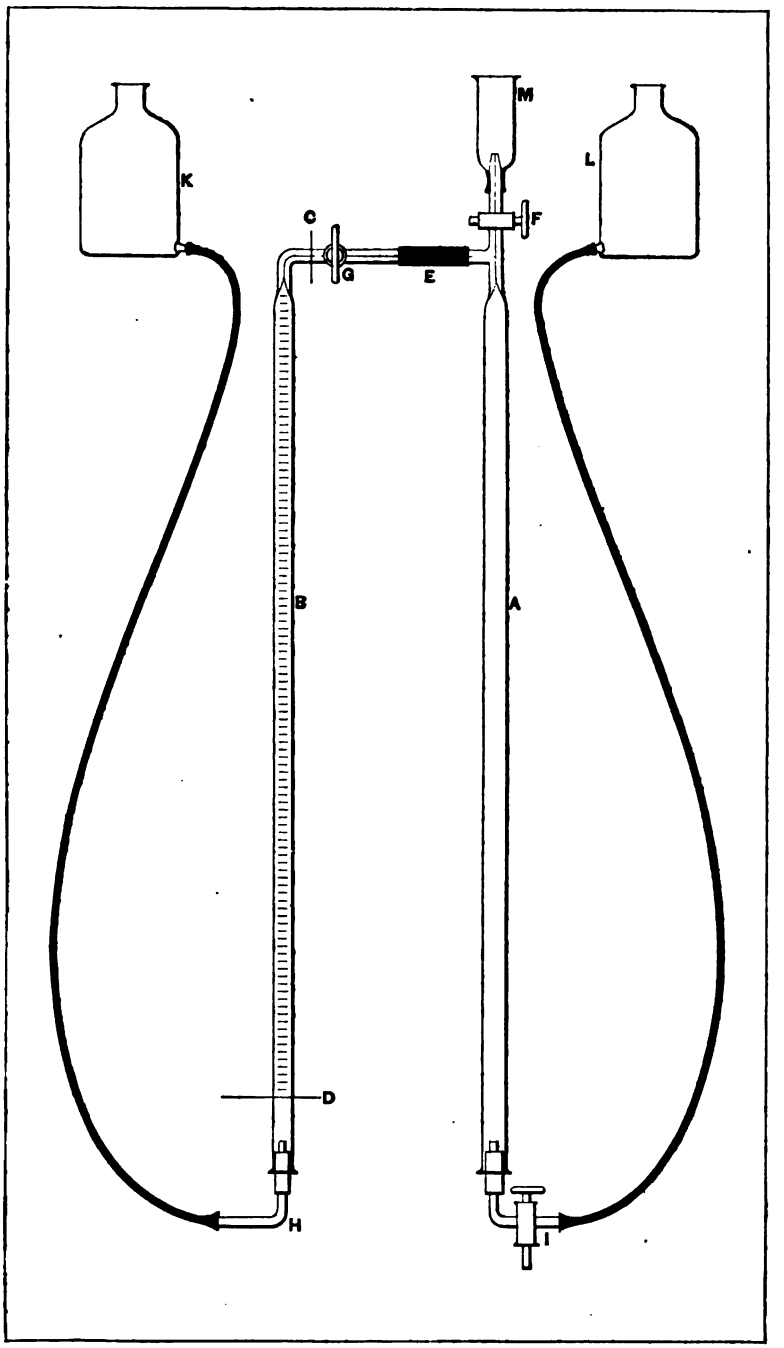

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A, and treated with potassium hydrate solution, to absorb hydrochloric acid vapors, before the final reading is made in B.

The treatment up to this point takes from twenty to thirty minutes, according to the amount of practice the operator has had with the apparatus. The gas residue still contains marsh gas, hydrogen, and nitrogen. By removing the funnel M, and attaching in its place a rubber tube communicating with an explosion eudiometer in a deep cylinder of water (both eudiometer and rubber tube being drawn full of water); a portion of the gas residue can be mixed with oxygen, exploded, and the contraction and the carbonic acid determined; the marsh gas and hydrogen being calculated by the usual formula. The nitrogen is found by the difference of the addition of the other constituents and one hundred.

The explosion tube is simply a tube like A, without the lower attachment and the lateral capillary tube above; the funnel M being retained, and two platinum wires are fused into the glass near the top, to give the spark for ignition. It is only necessary to clamp this tube down upon a piece of cork in a vessel of water during explosion, and adjust the water level in a tall cylinder of water when making the readings of contraction and absorption of carbonic dioxide.

The whole analysis can be readily completed in forty-five minutes, and with due care gives results that are practically correct. Analyses done with an Orsat apparatus, with great care, gave results almost identical with those obtained by the above described apparatus in one-fourth the amount of time.

The great advantage of this apparatus over the single tube method is, that the gas is not submitted to the action of the water used to wash out the chemicals, which I have found to reduce the volume of the illuminants by two per cent.

Of course this method will not compare with the methods of Bunsen and others, where very delicate readings and nice precautions are taken, but it gives very good results for rapid work, and answers every purpose in every-day practice in a gas or metallurgical works.

The water used in the apparatus should be of the same temperature as the room in which the analysis is made, and by careful handling little or no chemicals get into the bottle L.

When working in a warm place the tube B should be surrounded with a water-jacket, to prevent change of volume in the gas while under treatment.

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SANITARY CONDITION OF CITY HOUSES.

By F. N. OWEN, E.M. ('78).

It is proposed in this article to show briefly the machinery of the Board of Health in its "Complaint Department," and, secondly, to indicate the more important defects in the sanitary condition of city houses.

The city is divided into ten districts, each of which has a sanitary inspector assigned to it. These inspectors are physicians, and besides caring for disease and the general sanitary condition of their districts, are required to examine into the causes of nuisances complained of by citizens, and to apply such simple remedies as the placing of traps, ventilating of soil pipes, cleaning of privy vaults, repairing of leaders, and the like. They cannot, however (being physicians), be always expected to design or test sanitary appliances, or to take the time to investigate intricate cases, and it is in the study of such cases of house-drainage, sewerage, plumbing arrangements, water-supply, and ventilation, where radical changes are necessary to render a building safe to live in, that the engineer is employed. Quite recently the engineers have had districts assigned to them, and besides the work mentioned above, have the examination and supervision of the drainage, ventilation, and plumbing of new buildings, in accordance with the Plumbing Law passed in June of this year.

All inspections are made on notices of nuisance, which may be either complaints or requests for examination by owners. In the former case, orders for necessary alterations to abate the nuisance are served on the owners, and the premises are re-inspected, to see that the work has been properly done. Should the owner fail to comply with the order, the law is used to compel him to do so. In the latter case, after examination, a report in the form of a letter is sent to the owner, with recommendations, and the case is there dropped by the Board, with the feeling that if the person was intelligent enough to recognize the need of an examination, he certainly would also be willing to comply with the recommendations. In any case, only such repairs or alterations are demanded as will abate the nuisance, or, in the case of a dwelling, render it safe to live in.

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The engineer is obliged to confine himself to general principles in making his orders, as the Board cannot recommend any one particular appliance. Even the foot-trap, breaking the connection between the house-drain and the street sewer, has not been insisted on in old houses; but it is to be hoped that the time is not far distant when a measure equally important with the ventilation of soil-pipes (which is now required) will be insisted upon with equal rigor. The substitution of the "school sink," for the old-fashioned privy vault is insisted upon; and this is required, not because it is the best appliance that will ever be invented to meet the requirements of the case, but because it is the best one known at the present time. It has its disadvantages, such as tendency to freeze in winter and to become offensive if not regularly discharged, but when one comes to compare it with the foulness of the privy-vault—perhaps cleaned once a year, always leaking, and oftentimes built of stone, laid dry—the superiority of the school-sink is immediately seen. The writer remembers a case where the cellar of the building was used as a bakery, the oven extending from the rear wall out under the yard. In the yard was an old brick privy-vault, decidedly the worse for wear, leaking badly and saturating the sides of the oven with sewage; and yet the business was a thriving one, if one could judge by the variety of bread and cake displayed. In such a case, especially if it be in a densely-populated quarter, and is used by from ten to twenty-four families, a school sink is ordered in, the old privy-vault being first disinfected, cleaned, and filled with fresh earth. In making such an order, the engineer also sends to the Board a diagram, showing the location of the premises with reference to the surrounding houses. This is probably the most offensive nuisance that a health officer has to attack, but it is by no means the most dangerous to health.

	Brick drains.	Defective earthenware drains.	Privy-vaults.	Defective iron drains and pipes.	Insufficient ventilation of soil-pipe.	Leader used as ventilator for soil-pipe.	Soil-pipe not ventilated.	No traps.	Wet cellars.	Improper water-supply.	Air-duct to furnace open.
Tenement-houses.	6	16	33	8	5	3	8	5	32	2	..
Private dwellings..	1	15	..	8	14	4	18	11	27	9	2
Percentage.....	4.0	17.9	19.0	9.2	10.9	4.0	15.0	9.2	34.1	6.3	1.1

The above table gives a statistical record of one hundred

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and seventy-three cases, compiled from notes made by the writer, and shows the more glaring defects in the plumbing and drainage of New York houses, and the proportion in which they are distributed between tenements and private dwellings.

Brick drains are occasionally found among the older tenement houses, but owing to the rigorous measures of the Board they do not now exceed five per cent.

About fifty per cent. of the earthenware house-drains found in city houses are defective, owing to improper laying. Scarcely ever are they found bedded in cement, and generally have so slight a fall that, not cleansing readily, they become obstructed, the joints open, and the soil of the cellar becomes saturated with sewage. Ignorance on the part of plumbers has a great deal to do with the defective work in our houses. The writer knows of a case where a plumber (?) was found in a cellar, laying an earthenware drain with the spigot ends against the current! Under the good influences of the Plumbing Bill it is to be hoped that such work will no longer be possible.

Privy vaults are confined to the tenement-houses, and are fast giving way to school-sinks; and in small houses of three or four families, to water-closets.

It will be seen from the table that defective iron pipes are equally divided between private dwellings and tenements. This is due to the fact that the cause of defect is the same in both classes of building, viz.: improper caulking. Oftentimes pipes are found put together with cement, and even common mortar; and where the stack of soil-pipe is set in a chase, even the lead caulking is sometimes defective, from the difficulty of getting at the parts next the wall.

Insufficient ventilation of soil-pipes by pipes of smaller size is much more frequent in private dwellings than in tenements, and is probably due to the fact that when, some years ago, the theory of ventilation of pipes was taken up, it was not generally recognized that the full-size pipe was necessary; so that in private houses, where no organized inspection has been made, they are not brought to light till some complaint is made against the house.

The use of the soil-pipe for a rain-water leader is not very common (about four per cent.), but where such is the case, that part of the soil-pipe between the highest fixture and the roof is generally of tin and in a very defective condition, resulting

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in the part of the house through which it passes being permeated with sewer gas.

As with the insufficiently ventilated soil-pipes, so with the non-ventilated: the tenement-houses are better off than private dwellings by more than two to one, on account of the tenement-house inspection.

The same reason will account for the figures in the column of "No Traps," which has reference, of course, to sinks, basins etc., and not to water-closets.

The figures in the column for "Wet Cellars" refer to those that are not merely damp, but absolutely wet. About ninety per cent. of the cellars of New York are damp, if not wet. This is due to—

First.—Bad subsoil drainage, or none at all: none at all being the rule. The cellar floor, when cemented or flagged, is often graded to drain into a pipe, discharging into the house-drain, and provided with that worst of all traps—a "bell trap." When this is the case, the cellar is pretty certain to be in direct communication with the sewer.

Second.—Many cellars have no means for through ventilation. All have windows in front or rear, but probably not more than fifty per cent. have them at both ends, and in many cellars where such is the case they are closed so as to be practically useless. Others again are intended to be ventilated by an air-shaft extending to the roof, covered with an almost tight skylight, and provided with windows on each floor for the lighting and ventilation of water-closets or inside rooms. The result of this arrangement is that the cellar is more or less ventilated through these windows into the rooms. The inward current of air is perceptible at almost all such windows, especially during the winter, when the rooms are heated artificially.

Third.—I think it will be found, on careful investigation, that the dampness of cellars is largely due to percolation of surface water from the streets. The cellars, separated only by the house walls, and lying far below the level of the surrounding ground, act like huge catch-basins, taking the water as it percolates from the streets on the one side and the yards on the other. Owing to the unevenness of our street pavements, and the large spaces between the stones, the surface water

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readily settles into the ground, and finds its way into the cellars, to the detriment of the health of the inmates of the house.

The writer hopes to show, in a future article, the effect of damp cellars and cellar air upon the atmosphere of city houses.

The defects in water supply are mostly the direct communication of cistern wastes with sewers, allowing the gases to come in contact with the water and be absorbed by it.

The last defect mentioned in the table is only noticeable during the winter, when any openings in the air-duct feed cellar air into the furnace, to be heated and distributed throughout the house.

The limits of this article will not permit a more detailed description of the defects in the sanitary condition of city dwellings; but it is hoped that enough has been said to point out the more glaring ones, and assist the engineer in making his first attack on that great enemy—sewer-gas.

INACCESSIBLE DISTANCES IN SURFACE AND UNDERGROUND SURVEYING.

By H. S. MUNROE ('69) and J. W. DAVIS ('78).

By computing the latitudes and departures of a compass survey, or the co-ordinates of a traverse line, we have the means of supplying the bearing (or angle) and the length of *one missing side* of the polygon.

It frequently happens, however, that one station of the survey is inaccessible, and while we are able to take the bearings of all the courses, either directly or by back-sight, there remain *two unmeasured distances* to be supplied. Or, again, two sides, not adjacent, may be inaccessible, and, as before, we are able to determine the bearings, but cannot measure the lengths of these two sides.

The computations in such cases become complicated and tedious, and the present discussion has for its object the shortening of the work by means of formulæ of general application.

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Let A=bearing of the first unmeasured course, 7, 8, of the survey.

B=bearing of the second course, 8, 1.

C=angle between the courses.

L=total latitude, and

D=total departure of the two unmeasured courses, ob-

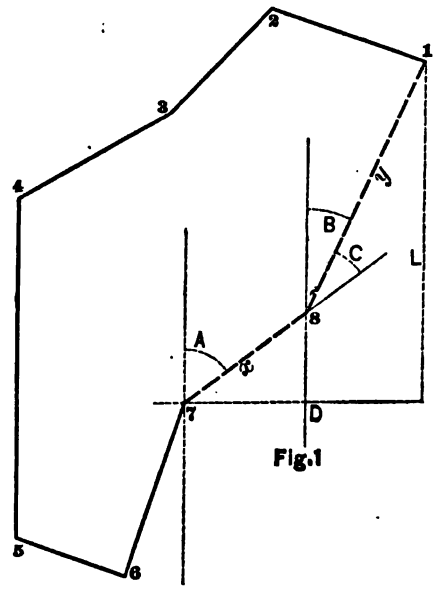

Fig. 1

tained by computing the total latitudes and total departures of the measured courses, 1, 2 to 6, 7.

Let x=length of the first course.

y=length of the second course.

We shall then have:

$$L=(\text{lat. } x+\text{lat. } y)=x \cos. A+y \cos. B. \quad (1.)$$

$$D=(\text{dep. } x+\text{dep. } y)=x \sin. A+y \sin. B. \quad (2.)$$

Multiply (2.) by cot. B.

$$D \cot. B=x \sin. A \cot. B+y \cos. B. \quad (3.)$$

Subtract (3.) from (1.) and solve with regard to x.

$$x=\frac{L-D \cot. B}{\cos. A-\sin. A \cot. B}.$$

Multiply both numerator and denominator by sin. B.

$$x=\frac{L \sin. B-D \cos. B}{\sin. (A-B)},$$

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or, substituting for sin. $(A-B)$ its value;

$$x=\frac{L \text{ sin. } B-D \text{ cos. } B}{\text{sin. } C}. \qquad (4.)$$

In like manner we have

$$y=\frac{D \text{ cos. } A-L \text{ sin. } A}{\text{sin. } C}. \qquad (5.)$$

By taking into account the signs of the different quantities, the above formulas (4.) and (5.) will solve any case that may arise, not only when the unmeasured sides are adjacent, as in the figure, but any two sides of the polygon, *e.g.*, sides, 7, 8, and 4, 5; provided only that the courses are not parallel, or nearly so, as 7, 8, and 3, 4.

The following table gives the proper signs in a convenient form for reference:

Signs of *L.* and *D.*	Signs of functions of *A* and *B.*
Lat. N.; L. is +.	N.E. bearing; sin. +, cos. +.
Dep. E.; D. is +.	S.E. bearing; sin. +, cos. —.
Lat. S.; L. is —.	S.W. bearing; sin. —, cos. —.
Dep. W.; D is —.	N.W. bearing; sin. —, cos. +.

To find value of C.:

$C=(A-B)$ if courses are in same or in opposite quadrants.

$C=(A+B)$ if the courses are in adjacent north or adjacent south quadrants.

$C=180°-(A+B)$ if courses are in adjacent east quadrants or adjacent west quadrants.

EXAMPLE:

Course.	Bearing.	Dist.	Latitude.		Departure.	
			N.	S.	E.	W.
D 321 *v* to *D* 322 *b*.	East.	48.	....	....	48.00	....
D 322 *b* to *D* 322 *a*.	N. 71½ E.	5.18	1.64	....	4.91	....
D 322 *a* to *D* 322 *s*.	S. 13½ E.	34.	....	33.13	7.65	....
D 322 *s* to *D* 322 *y*.	South.	12.	....	12.00	....	....
D 322 *y* to *M* 3710 *a*.	S. 89° 10′ W. (*A*)	(*x*)	....	....	....	....
M 3710 *a* to *D* 321 *v*.	N. 80° 20′ E. (*B*)	(*y*)	....	....	....	....
			1.64	45.13	60.56	

$L=+43.49$ $D=-60.56$

Here the point *M* 3710 *a* was on one side of a deep ravine, and the character of the ground on the other side made it impossible to measure a straight base, the ends of which would be available

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for triangulation. Sights on the inaccessible station M 3710 a were therefore taken from the end stations of a short traverse.

From the above table we find that, in order to balance the latitude and departure columns, the total latitude of the two unknown courses must be 43.49 North, and the total departure 60.56 West. Substituting in the formulas—

$$x = D322\,y \text{ to } M3710\,a = \frac{43.49 \times sin.\,80^\circ\,20' + 60.56 \times cos.\,80^\circ\,20'}{sin.\,8^\circ\,50'}$$

$$y = M3710\,a \text{ to } D321\,v = \frac{60.56 \times cos.\,89^\circ\,10' + 43.49 \times sin.\,89^\circ\,10'}{sin.\,8^\circ\,50'}$$

Solving, with the aid of logarithms,

$$x = 345.41 \qquad y = 288.91$$

As a check, the latitudes and departures of the two sides should be computed, and the survey closed as below.

	Bearing.	Dist.	Latitude.		Departure.	
			N.	S.	E.	W.
D 321 v to D 322 b..	East.	48.	....	....	48.00	....
D 322 b to D 322 a..	N. 71½ E.	5.18	1.64	....	4.91	....
D 322 a to D 322 z..	S. 13½ E.	34.	....	33.13	7.65	....
D 322 z to D 322 y..	South.	12.	....	12.00	....	....
D 322 y to M 3710 a.	S. 89° 10' W.	345.41	....	5.02	....	345.37
M 3710 a to D 321 v.	N. 80° 20' E.	288.91	48.51	....	284.81	....
			50.15	50.15	345.37	345.37

The following example will illustrate an application of the formulæ to a problem in mine surveying:

Course.	Bearing.	Dist.	Latitude.		Departure.		Total Latitude.	Total Departure.
			N.	S.	E.	W.		
a–b..	S. 30° 10' W.	172.86	....	149.45	....	86.87	−149.45	− 86.87
b–c..	S. 19° 42' E.	2.83	....	2.66	.96	....	−152.11	− 85.91
c–d..	N. 73° 43' E.	111.52	31.27	....	107.05	....	−120.84	+ 21.14
d–e..	N. 1° 08' W.	25.22	25.21	....	....	.50	− 95.63	+ 20.64
e–f..	N. 89° 32' E.	51.25	0.42	....	51.25	....	− 95.21	+ 71.89
f–h..	S. 0° 29' E.	206.43	....	206.43	1.75	....	−301.64	+ 73.63
h–i..	N. 57° 14' E.	528.01	285.77	....	443.99	....	− 15.87	+517.62
i–j..	N. 30° 0' W.	416.33	360.55	....	....	208.17	+344.68	+309.46
j–a..	S. 41° 55' W.	463.22	....	344.68	....	309.46	0.	0.
			703.22	703.22	605.00	605.00	.....	.....
e–ea .	N. 1° 8' W.	23.46	23.46	....	....	.46	−72.17	+ 20.17
f–g..	N. 33° 16' E.	102.37	85.59	....	56.15	....	− 9.62	+128.04

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The courses *b–c* to *f–h* of the above survey are supposed to be in the gangways and shafts of a coal-mine. The remainder of the survey is over the surface. The side, *j–a*, follows a property line. The gangways, *e–ea* and *f–g*, are to be driven up to this property line, or its prolongation, and no farther. Required, the distances from *ea* and from *g* to the line.

$$x = \frac{-72.17 \; sin.\; 41^\circ\; 55' - 20.17 \; cos.\; 41^\circ\; 55'}{-sin.\; 43^\circ\; 3'}$$

$= 92.62$, the distance from *ea* to the property line.

$$x = \frac{-9.62 \; sin.\; 41^\circ\; 55' - 128.04 \; cos.\; 41^\circ\; 55'}{-sin.\; 8^\circ\; 39'}$$

$= 676.23$, the distance from *g* to the property line.

It is evident that these formulæ will have many applications besides those indicated. For example, the intersections of the side lines of United States patent surveys may be calculated, if necessary, with far greater accuracy than they can be determined by plotting. The formulæ could also be used to determine the lengths of the sides of crooked claims.

TO READ A COMPASS TO MINUTES.

In using the compass for triangulation, as in the first example, it will be necessary to read the bearing to within a few minutes. When the compass reads to degrees only, and has no vernier, this may be done by means of a graduated card, fastened to a sight-pole at some convenient distance from the instrument. At ten feet a card two inches long subtends almost exactly one degree, and therefore one-third of an inch equals ten minutes. Place the zero of this graduated card accurately in the line of sight, and then turn the instrument till the needle coincides with nearest degree-mark. On sighting again at the card the number of divisions will give the minutes with sufficient accuracy for most purposes.

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ON ELECTRO-SILVER-PLATING.

By CHARLES A. WITTMACK ('82).

History.—After the simultaneous discoveries of Jacobi,[1] Jordan, and Spencer, in 1839, of practically applying the galvanic current to the precipitation of copper upon the surfaces of useful articles, ornaments, and the like, this art made rapid advancement. The great uses especially that the noble metals would find, if they could be similarly deposited, was at once recognized. In 1840, Messrs. Wright and Parkes, of Birmingham, experimented on solutions of these metals; but as there are only very few solutions that give good deposits, they were at first only partially successful.[2]

During his studies, Mr. Wright met with a passage in "Scheele's Chemical Essays," in which there was a description of the solubility of the oxides and cyanides of gold and silver, which was as follows: "If, after these calces (*i.e.*, cyanides of gold and silver) have been precipitated, a sufficient quantity of the precipitating liquor be added in order to redissolve them, the solution remains clear in the open air, and in this state the aërial acid (*i.e.*, carbonic acid) does not precipitate the metallic calx."[3]

Mr. Wright, thinking that this solution might be suitable for his purposes, dissolved some chloride of silver in an aqueous solution of ferrocyanide of potassium. Upon connecting the wires of the battery he obtained a beautiful white coat of silver, adhering firmly to the metal underneath, which was, therefore, entirely suited for plating. He afterward made use of a solution of pure AgCy, with a little free KCy, and this has been found to be the most convenient solution up to the present time. This process was patented by Messrs. G. R. & H. Elkington, in 1840—these gentlemen being proprietors of a large establishment in Birmingham, England. They paid Mr. Wright a royalty of one shilling for every ounce of silver de-

[1] Athenæum, May 4, 1839.

[2] Gore: Electro-Metallurgy, p. 19.

[3] Scheele's Chem. Essays, pp. 405 and 406.

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posited. Elkington also took out a patent on the same process in France, and this was quickly succeeded by a patent taken out by De Ruolz for solutions of gold in cyanide of potassium, etc.[1]

During the year 1840, Professor De la Rive, of Geneva, described a process used by him in 1829, by which he plated wires of platinum with silver and gold. He, therefore, was the pioneer in the art of gold- and silver-plating.[2]

Old methods of silver-plating.—Before the process of precipitating silver by means of the galvanic current was known, two essentially different methods were employed for coating articles with this noble metal. The first and oldest of these was as follows:

Silver was rolled out in very thin plates, pasted upon the articles to be plated, and then polished with steel or agate. If the covering of silver was to be at all durable, it was necessary to lay forty to fifty of such plates upon one another.

This process was superseded by one in which a comparatively thick sheet of copper was laid upon a thin sheet of silver, and having first been warmed, they were passed through heavy steel rollers. The silver and copper united, and could be rolled out to a thickness suitable for plating. These silver-plated sheets of copper were now worked by means of heavy steel dies. The simplest articles, as, for example, a candlestick or a tea-pot, were soldered together out of many small pieces. The handles, ornaments and other small parts, were also stamped out of silver-plated copper, and afterward, the interior having been filled with another metal, were soldered to the large parts. The dies were very expensive, being made of the best steel. The plated articles became unsightly and useless as soon as the silver was worn off, there being no process by which it could be replaced.

Preparation of cyanide solution.—The solution which has been found the most suitable is that of the double cyanide of silver and potassium, containing a little free cyanide of potassium. This solution is made up by use of the battery or by chemical means. The former operation is conducted as follows.[3]

Dissolve 75 grms. of cyanide of potassium in 10 litres of

[1] Encyclopédie Roret: Galvano-plast., tome ii., p. 114.

[2] Vide De la Rive's Treatise on Electricity, vol. iii., p. 546.

[3] Muspratt: Theor. and Prakt. Chem. Ed. 1875, pp. 49, 50.

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water, introduce a porous jar into the solution, and fill it up with the same solution. Place a copper or iron cylinder into the porous jar, and connect this with the zinc pole of the battery. By using a Wollaston battery, consisting of four pairs of plates (the zinc plate being 15 ctm. broad), a silver solution of sufficient strength can be obtained in a few hours. By weighing the silver slab before and after the operation, the amount of silver dissolved can be ascertained.

Gore gives the following method for making the solution by chemical means.[1] To four parts of grain silver add six and one-half parts by weight of strong nitric acid, and one part of warm water, being careful to maintain a moderate and uniform action until all is dissolved. Evaporate nearly to dryness, to expel the excess of acid. The resulting nitrate of silver is dissolved in water, in the proportion of one-half gallon to each ounce of silver used. Just enough of a solution of cyanide of potassium (2–3 parts of the salt in 20–30 parts of water) is now added to precipitate all the silver. Allow the precipitate to settle and decant off the supernatant fluid, wash by decantation 5–6 times with cold water. Now add a sufficient amount of a solution of 6–8 parts of cyanide of potassium in 20 parts of distilled water, so as to barely dissolve the precipitate, and finally 3 parts more of the salt are added to furnish the free cyanide. Any residue should be removed from the solution and afterward treated for the silver it may contain. It is necessary to have free cyanide of potassium, because in working the solution insoluble cyanide of silver is formed at the anode, and requires free cyanide of potassium to combine with it and form the soluble double cyanide.

Cleansing of the articles to be plated.—In order to obtain a good coating of silver, the articles to be plated should be most carefully cleaned and prepared. Two processes are now in use for this purpose, known as the dry and the wet process.

The dry process consists in cleaning the article with fine sand or emery cloth, and then polishing it with pumice finely ground, applied with a wire brush. The polished surface ought not to be touched with the hand, as it would become greasy as well as oxidized in consequence. This mode of cleaning articles to be plated is but seldom employed, as the wet process is now generally used.

[1] Elect.-Metall., pp. 156, 157.

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The wet process consists in dissolving away all grease and the like by dipping the articles in potash solution.[1] Then the process varies according to the nature of the impurities on the surface and the metal underneath. Articles of tin, lead, and Britannia metal may be placed in the depositing-vat immediately after being dipped into the potash solution and washed. Iron articles are cleaned in a solution of dilute sulphuric acid (1 : 20 parts) in which a few ounces of zinc have been dissolved. A number of liquids are required for copper, brass, and German silver, viz.: first, strong nitric acid, then a mixture of dilute sulphuric and nitric acid, with a little hydrochloric acid, and lastly, a still more dilute solution of the same acids.

Batteries.—For generating the electricity many batteries are made use of, Smee's battery being recommended as one of the best.[1]

About eighteen cells of this battery are generally made use of, the slabs of metal being 60 ctm. long and 22–25 ctm. broad. Magneto-electric machines have also come into use, and are now employed in the larger electro-plating establishments. The most celebrated of these are Wilde's,[2] Gramme's,[3] and Siemens' and Alteneck's.[4]

Thermo-electricity is also used. There are two varieties of the thermo-electric pile which are very efficient—those of Nöe, of Vienna,[5] and Clamond,[6] of Paris.

Plating.—The articles to be plated are suspended from the cathode or minus pole in the silvering solution, while a plate of silver is hung from the anode. As it often happens that silver deposits irregularly on a surface, giving it a striped appearance, arrangements have been devised in order to impart a slight motion to the articles suspended from the cathode and anode. Articles which are to be replated must first be thoroughly cleaned from silver before a new coating is deposited upon them. This may be done by mechanical means or by suspending the article from the anode of the battery and putting a plate of silver at the cathode. The silver will separate from the article and deposit upon the silver plate at the

[1] Muspratt: Pract. Chem., p. 46.
[2] Ganot: Physics, p. 812; Philos. Mag., June, 1873.
[3] Electrical News, vol. i., p. 54; Ganot: Physics, p. 816.
[4] Ibid., vol. i., p. 226. [5] Teleg. Journal, vol. i., p. 12.
[6] Ibid., vol. iii., p. 157, and vol. iii., p. 319; Ganot: Physics, p. 844.

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cathode. By simply changing the poles, the silver can be deposited upon the article to be plated. To deposit silver upon copper, brass, or German silver, they are first coated with a thin film of mercury, causing the silver to adhere more firmly. This operation is known as quickening. The nitrate or cyanide of mercury is the salt generally used. A great part of the better qualities of plated ware is covered with a very thin film of palladium, to prevent the action of sulphuretted fumes on the silver.[1]

Special Methods.—Among the various special methods may be mentioned that of Boettger[2] for plating cast-iron with silver. He makes use of nitric acid (specific gravity 1.2) for cleaning the cast-iron, and then uses a silver-bath of the following composition: 1 part nitrate of silver is dissolved in 16 parts of distilled water; this is precipitated and redissolved by adding 2 parts of cyanide of potassium, and then the whole is diluted by adding 48 parts of water containing 1 part of chloride of sodium, the battery to be used consisting of two to three cells, moderately strong. Towle[3] invented a process for silver-plating the interior of lead-pipes. To electroplate over soft solder, clean the articles well with caustic potash, dip them in red fuming nitric acid, and then wash thoroughly. The soldered portion is then dipped in a very dilute solution of cyanide of mercury and potassium. They are then rinsed again, and placed in the plating-vat.[4]

Bright silver surfaces may be obtained by the addition of a small quantity of bisulphide of carbon to the silvering solution.[5] Silver so deposited is not pure, as sulphur may be found in it by dissolving it in nitric acid and determining the sulphuric acid by the ordinary method. If a bright silver deposit is not immediately immersed in boiling water and left there for some time, it will turn black. Silver is often deposited from ordinary cyanide solution, and then made shining by mechanical means.

Finally, a few remarks on the ornamentation of silver-plated articles might here be appropriate. If it is desired to obtain a "dead appearance" upon a surface of silver, deposit an exceedingly thin film of copper upon it from a solution of the sulphate, and upon this again a thin film of silver. "Oxi-

[1] Wagner: Chem. Tech., p. 116.
[2] Dingl. Journ., 191, p. 250.
[3] Ibid., 201, p. 49.
[4] Gore: Elect.-Metall., pp. 164 and 321.
[5] Ibid., pp. 167, 168.

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dized silver," as it is termed, although it does not consist in coating a deposit of silver with its oxide, as its name would indicate, is made by dipping silver articles into a hot solution of platinic chloride, the platinum separating in a finely divided state, and giving the silver a tint varying from a steel-gray to a deep bluish black. "Nielled silvering" consists in inlaying the surface with sulphides of silver, previously prepared, and causing them to adhere by heating them in a muffle.

Silver surfaces are often gold-plated. This is done by covering those parts not to be plated with copal varnish, which protects them and allows the gold to adhere to the bare parts only.

THE MARQUETTE IRON REGION.—I.

By THE STUDENTS[1] OF THE SUMMER SCHOOL OF PRACTICAL MINING, LAKE SUPERIOR, 1881.

GEOLOGY.

THE "Marquette Iron Region" is situated within the county of Marquette, a central county of the upper peninsula of Michigan, bounded northeast by Lake Superior, and southwest by Wisconsin, and drained by the Escanaba, Michigamme, and Mequacumecum Rivers and other streams.

The rock-formations of the district are embraced under three systems, which, beginning with the oldest and lowermost, are: 1. The Laurentian, or granitic rocks; 2. The Huronian, or iron-bearing rocks; and 3. The Lower Silurian, or sandstone rocks.

The Laurentian System, the oldest on the peninsula, extends southeast to Lake Ontario and along the north side of the St. Lawrence to the sea. The rocks, with few exceptions, are metamorphic or crystalline. They include granite, gneiss, and some mica-schist. Besides these there are quartzyte, dolomitic conglomerate, pyroxene rocks, syenyte and dioryte; certain feldspathic rocks, especially labradorite, and occasionally hypersthenyte, consisting of cleavable labradorite with

[1] Being extracts from the memoirs required of each student of the Summer School.

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true foliated hypersthene. Apatite, mica, graphite, and other minerals occur, disseminated through the crystalline limestone. Chloritic gneiss of all varieties of color especially abounds. The beds of rock constituting the system are usually tilted at high angles, the whole series having been upturned and flexed, broken and displaced, until little evidence of the original deposition—in horizontal strata—remains.

The Huronian System.—The rocks of this system lying unconformably upon the Laurentian rocks, thus showing their later origin, consist of dioryte, chloritic schist, quartzyte, beds of jasper, and layers of slates and shales among which are found extensive beds of iron-ore. The rocks of this series have for the most part been tilted, the inclination varying from a few degrees to vertical. They have been raised into folds or crumpled into groups of irregular flexure, forming a series of irregular synclinal troughs, separated by anticlinal domes or crests, and at times, by reason of their upturned edges, imparting to the district its topographical configuration. The prevailing rocks are diorytes and dioryte schists, locally termed "greenstones" or "traps." They are hard, compact, heavy, tough, medium-grained rocks, from dull light green to blackish green in color, and consist essentially of amphibole, showing in minute glittering blades or facets of greenish gray or grayish white feldspar, with quartz, calcite, and mica—the latter disseminated throughout the mass in minute scales, usually of a brown or brownish yellow tint. In these rocks there appears to be no well-defined lamination. No marked arrangement of the minerals in parallel planes can be discerned. All appearance of stratification is lost, and hence it is often difficult to determine the dip of the beds except with the aid of the schists and slates and other more plainly stratified members of the same group. Associated with the hard ores of the region is a soft, compact, homogeneous, fine-grained magnesian rock, chloritic or talcose in character, and consisting of grayish white quartz in minute granules, of scales of olive-green or blackish green chlorite, and occasionally of crystals of amphibole and particles of feldspar and mica. It is frequently found overlying the ore-bed, and this being the case, the hanging wall in places, in many of the mines, is rendered exceedingly dangerous. As a rule, it is considered better policy to take down this rock and reach the solid quartzyte, which, throughout the region, is the true

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hanging wall of the ore. This upper quartzyte, which differs from the lower quartzyte bed in that the latter is often calcareous, is a rather brittle, hard, tough, compact rock, of moderate fineness of texture, possessing usually a light or dark gray color, sometimes inclining, however, to reddish or light chocolate-brown, from the presence of ferric oxide. It is massive, rarely showing any signs of bedding, and at times merges into jaspery quartz conglomerate composed of pebbles of white or glassy quartz and jasper.

In addition to these three classes of rock, the identifying of one or more of which in any locality in the region is so essential a step to the discovery of the ore-deposit itself, there are also found more or less regular beds of mica-schist, containing crystals of staurolite and andalusite; beds of anthophyllitic schist, a rock composed of anthophyllite and quartz, usually with minute grains of magnetite, giving it its customary brown or brownish black color, and often showing the presence of manganese; also beds of a black or bluish black shale, hard, compact, and fine-grained containing a small percentage of carbonaceous matter.

Major T. B. Brooks, in his report, "Geological Survey of Michigan, 1873," gives a complete classification of the rocks of the Huronian System, making use of Roman numerals, I. to XIX., to designate the different beds recognized by him. I., II., III., and IV. are beds of siliceous ferruginous schist, alternating with chlorite schists and dioryte, the relations of which have not yet been fully determined. V. is a quartzyte, sometimes containing marble and beds of argillyte and novaculyte. VI., VIII., and X. are beds of siliceous ferruginous schist. VII., IX., and XI. are diorytic rocks, varying much in character. XIII. is the bed which contains all the rich specular and magnetic ore deposits, associated with mixed ore—the so-called "jasper," and with magnesian schist—"chlorite." XIV. is a quartzyte, often conglomeritic; XV. is argillyte or clay-slate; XVI. and XVII. somewhat uncertain, the former containing some soft hematite deposits; XVIII. is doubtful; XIX. is mica-schist, containing staurolite, andalusite, and garnets.

The Lower Silurian System is represented on the upper peninsula by the Potsdam and calciferous sandstones, and by various deposits of limestone in rather small beds. The sandstones vary in color from almost white in the upper to a red in the middle and lower formations. Many of the beds furnish an excellent stone for building purposes.

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The several beds of the Huronian System—the iron-bearing rocks—with numerous minor folds and corrugations, form a broad, synclinal basin, which extends from Marquette westward to a distance of nearly fifty miles. The general trend of the formation, which, however, is subject to great local irregularities, is east and west, and most of the numerous iron deposits scattered throughout the district are contained in strata forming minor folds, which conform in direction to the axis of the main synclinal trough.

The iron region may be divided into the (1) Negaunee, (2) Michigamme, (3) Escanaba, and (4) L'Anse districts. These divisions may be conveniently carried still further, by a subdivision of the Negaunee district into the Cascade range, Negaunee hematite mines, Ishpeming group, New England and Saginaw range. The Michigamme district may, in like manner, be divided into the Washington, Spurr, and Magnetic ranges, and the Republic basin.

W. A. SHUMWAY.
D. B. TOUCEY.

HISTORICAL.

The first information of the existence of iron in this region is largely due to the Government surveyors, who, under the direction of Dr. Douglas Houghton, the first State Geologist of Michigan, were required to combine geological and topographical observations with their linear surveys, and to record the results upon their maps. Dr. Douglas Houghton first visited Lake Superior in 1830, though it was not until the following year that his work of systematic scientific exploration was begun. His report to the Legislature in 1841, after ten years of careful and active toil, presented the results of his labors up to that period in so able a manner that the attention of the East became directed to the upper peninsula as promising to richly reward further exploration and the investment of capital.

At the time of his report (1841), Dr. Houghton had no knowledge of the existence of iron-ore in large quantities, his geological investigations in the interior of the country having been entirely confined to the copper region. In 1841 the State of Michigan, owing to a financial crisis, withheld its annual appropriation for the geological survey, which, for twelve years had been so successfully conducted. Dr. Houghton then

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sought assistance from the General Government, and succeeded in having an additional allowance granted for the linear surveys. He himself took the contract for the completion of the surveys, the Hon. William A. Burt, United States Deputy Surveyor, being placed in charge of the field work, receiving as compensation the whole amount of the additional appropriation. In the summer of 1844, Mr. Burt, in company with a party of United States Surveyors, first noted the existence of iron-ore in place, while running the east line of T. 47 N., R. 27 W., being led to the discovery by observing, by means of the solar compass, remarkable deflection of the needle, amounting to over eighty degrees from the normal. A search on either side of the line revealed the presence of outcrops of magnetic iron-ore, specimens of which were secured and described by him in the report accompanying the linear surveys. This was unquestionably the first discovery by white men of iron-ore on Lake Superior, though the credit of having given to the world information of the existence of workable deposits of merchantable iron-ore was reserved for S. T. Carr and E. S. Rockwell, who were members of a party organized under P. M. Everett, at Jackson, Mich., in June, 1845, for the purpose of exploring the mineral regions on the south shore of Lake Superior. Their discovery, due in large measure to the guidance of an Indian chief named Manjekijik, was afterward developed, and is now known as the Jackson Mine, situated in the city of Negaunee. The first ore was taken from the mine in the summer of 1846, and the first regular shipment, of about 5,000 tons of ore, was made in the year 1856.

The early history of the operations of the company up to 1860 is but a record of changes, disappointments, and financial embarrassments. In 1861 a new board of directors was chosen, and in 1862 the Jackson Company declared its first dividend, since which time the mine has yielded nearly two millions gross tons of ore.

The above brief sketch is important only as showing the progress in the development of the Marquette region, the Jackson mine being the first opened, and its history that of many other companies. It is interesting to note that the purpose of the company at the time of its organization was to mine ores of more valuable metals than iron.

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CHARACTER OF THE ORE-BODIES.

The Champion Iron Mine is located about eighteen miles from Marquette, on the line of the Marquette, Houghton and Ontanagon Railroad, and lies between the Michigamme and the Republic Mines.

The deposit lies in one of the minor folds of the iron-bearing rocks, which outcrop at this place on opposite sides of the basin, about three miles apart. The position of the ore deposit is very nearly due east and west, and is nearly vertical, having a dip of 83° to the north.

The ore-bed consists of a number of "lenses" or separate masses of iron-ore of an irregular flattened form. These lenses are found slightly overlapping, and the west edge of one lens is usually found to the north side of the one next following.[1] This position of the lenses is shown in the accompanying plate, Figures 1, 2, 3 being horizontal sections through the ore-bodies at the surface and at different levels.[2] The ore-bodies pitch about 45° to the west.

The lenses are distinguished in this mine as those forming the south belt and those of the north belt. Shafts Nos. 2 and 3 are sunk in the north belt, and Nos. 4 to 7 in the south belt. South of these two main groups are several smaller deposits of very irregular shape, known as the "Chimney," the "Old Man's Deposit," etc.

The foot-wall of the ore-bed is dioryte and the hanging wall quartzyte. Chloritic schist occurs in large quantity between the lenses and forms in nearly every case the rock-bedding under the different lenses. The true foot-wall (dioryte) of the ore-bed has not yet been reached in the underground workings, and the thickness of the bed is estimated to be from 100 to 150 feet. Jasper[3] occurs mostly on the foot-wall side of the lenses and back of the chlorite-schist. In one case it was noticed to be in contact with the ore and forming a dividing wall between two adjacent deposits.

[1] And the lower edge of the lens seems usually to lap the one below on the hanging wall side, in this case also to the north.—ED.

[2] These "lenses" have shapes somewhat like pea-pods, and usually occupy inclined positions in the ore-bearing stratum, the longer axes more or less parallel, dipping to the north with the enclosing rocks, but by reason of their diagonal position having also a "pitch" toward the west.—ED.

[3] The term "Jasper" is in the iron region usually applied to a low grade siliceous mixed ore, consisting of thin alternating laminæ of ore and jasper.—ED.

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SECTIONS OF ORE-BODIES, MARQUETTE IRON REGION, MICH.

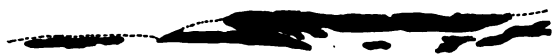

FIG. 1.—Champion Mine—Horizontal section at 5th level, 300 feet from surface.

FIG. 2.—Champion Mine—Horizontal section at 1st level, 60 feet from surface.

FIG. 3.—Champion Mine—Horizontal section of ore-bodies at the surface.

FIG. 4.—Jackson Mine—Horizontal section through ore-bodies at the tunnel level.

FIG. 6.—Dickerson Mine, N. J.

Horizontal section at 175 ft.

FIG. 5.—Jackson Mine—Pit No. 7.

SCALE OF SECTIONS, 1 to 5,000.

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The shafts are sunk near the hanging wall of the ore-bearing stratum, and the exploration and development of the mine proceed at the same time toward the foot.

If the theory of the accumulation of the ore by the transporting and oxidizing action of water be accepted, it may be assumed that we have in these deposits a number of beds of former lakes which stretched along in a broken series from the Michigamme to the Republic Mine.

The peculiarity of position of the ore-deposits seems to show that at one time a neighboring water-shed gave the surface-drainage this east and west direction, while the local topography caused the water to accumulate in shallow lakes, in which the conditions were favorable for the deposition of iron. The abundance of small streams and of granitic rocks, and the similar position of many small chains of lakes in the surrounding country at the present time, seem to indicate conditions similar to those which prevailed during the age of the iron deposits. The ore produced under such circumstances is limonite, which subsequent metamorphism transforms into magnetites and hematites.

The very low percentage of phosphates and earthy sediments in the ore of this mine seems at first sight irreconcilable with a sedimentary origin, and the very remarkable accumulation and overlapping of the lenses in the ore-bed requires further careful study to construct a theory which shall account for all the facts observed. The overlapping and grouping of the lenses may, perhaps, be attributed to the occurrence of cycles in the history of the formation of the ore-beds, due to disturbing causes, whose effect was to suspend for a time the deposition of the iron-ore and to shift slightly the position of subsequent accumulations.

The Champion Iron Mine is a good one to study in this connection, from the general simplicity of the deposits and lack of complication by serious faults.

Finally, it may be remarked that one is struck with the fact of some agency at work in sorting the materials of the rocks on the hanging wall side of the deposit. These rocks above the iron-ore consist of quartzyte (hanging wall) overlaid by beds of varying thickness of clay, slate, and talcose-schist. Imagining a section of one of these ancient lake-beds, the foreign substances brought in by streams would consist of the detrita of the surrounding rocks, so that in the order of their

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specific gravities we should have first the iron as a deposit on the bottom, covered by quartz-sand in a more or less fine degree of comminution, forming the jasper and quartzites of the present deposit. Above the sands we should find deposits of clays, fine mud, and silt, which would form the clay-slates and talcose-schists of the present day. The forces most actively at work in making these separations may be credited to the wind and rain acting through long periods of time upon the surface of the lakes, and, while they lasted, tending to communicate through the water a sort of jigging motion to the sediments at the bottom.

C. Q. PAYNE.

The above suggestions of Mr. Payne open up a very interesting train of thought. That the mechanical action of water has played in certain cases a very important part in the formation, or rather in the concentration and purification of these deposits of very rich ores, can hardly be doubted. This mechanical action, however, is probably more nearly akin to buddling than to jigging. To concentrate by jigging, we must have very positive upward movements of the water and the sands, followed by short periods of rest, to allow the quick-falling grains of heavy mineral to separate themselves gradually from the slow-falling grains of quartz, clay, etc. This positive upward movement could hardly be effected by any agency acting on the surface of the water. Again, we must provide for the removal of the lighter materials from the bed of the lake, and that this removal shall be faster than the inflow of earthy matter at the upper end, in order to have any concentration or enrichment of the iron deposits at the bottom. In the case of a lake fed by streams, however, the water would flow out at the lower end clearer than it entered above—the lake acting as a settling-basin. It is clear, then, that the deposit formed at the bottom would be no richer, on the average, than the sediment flowing in. There might be local patches of concentration, due to wave-action on the shores of the lake, or to the deposition of heavy sediment at the mouths of small streams, and to similar causes; but the deposit would be like the immense beds of "mixed ore" found in the iron regions, and would consist of low grade ore interstratified with beds of sand and clay. Occasional storms might stir up the lake, so that the outflow would for a time be muddy, but this would

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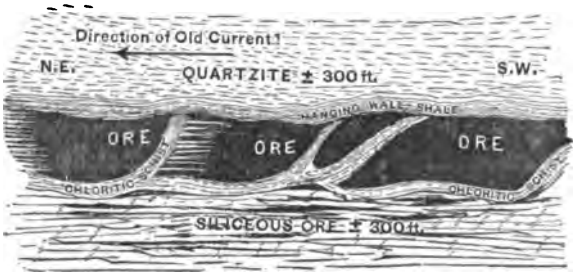
be only temporary, and any slight concentration due to this cause would be nullified by the constant inflow of fresh sediment above.

We must look, therefore, to the action of running streams, if we would account for the occurrence of these masses of pure ore by any theory of concentration by water; that, is to a method of enrichment similar to the operation of buddling in the dressing-works. The most common form of buddle consists of a rectangular box, with low sides and tilted up somewhat at one end. A stream of water is arranged to flow through the buddle, the bottom of which is carefully levelled crosswise so that the stream may spread uniformly in its passage across the whole width of the box. The sands to be concentrated are fed into this stream of water. The heavier sands settle at once near the head of the buddle, while the lighter sands are carried toward the lower end. The very finest and lightest



Cross-section through ore-bodies—fourth level, Edwards Mine.

clays and sands do not settle at all, but pass through the box and out at the lower end. When the buddle is full the concentrated heavy mineral is dug out from the head or upper end of the box; a product of medium richness, "middlings," from the middle; and a poorer grade of mixed ore and gangue from the lower end.

The above section, constructed from data contained in Major Brooks' report,[1] and based on a careful map of the Edwards Mine, by Mr. A. Kidder, of Marquette, throws considerable light on the probable origin of many similar compound and complex lens-formed deposits. The four lenses shown in the section, with the intercalated seams of chloritic schist, suggest in their arrangement the false-bedding or cross-

[1] Geol. Survey of Michigan, vol. i., pp. 122–125; also Plate XIX., opposite page 278, and Map No. VIII. of Atlas.

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bedding sometimes found in sandstone strata. This false bedding is due to the action of running water depositing sediment in successive layers on a sloping bank. For example, during and after a freshet, any stream heavily charged with sediment will deposit beds of sand having this structure in the pools and wider portions of its bed. Magnetic iron sand derived from the erosion of metamorphic rocks, or limonite sand from bog-ore deposits above, would be deposited even where the current was swift, the stream forming a natural buddle, all the sand and clay being carried onward and only the pure ore left. As the freshet subsides and the velocity of the current becomes less, a thin film of mud will be deposited on the iron ore, forming when metamorphosed the chloritic and talcose schists and soapstones. The iron sand thus deposited by the stream would resist erosion in subsequent freshets, especially in the richer and heavier portions at the head of the natural buddle. The lighter "tails" of the deposit would generally be swept away, forming a place for a new deposit beyond and overlapping the old one. The ore-bodies of the Edwards Mine, above figured, conform in a striking manner to all the characteristics of such a deposit, not only in their position and arrangement, and the overlying and intercalated beds of schists, but the pure ore is in each case at the head, and the mixed ore at the tail of the natural buddle, as indicated in the sketch. In the right hand lens the mixed ore seems to be absent, the tail of the buddle having been removed by erosion before the other ore-bodies were deposited. Another ore-body, not shown in the sketch, shows the same arrangement, the rich ore at the head and the mixed ore in its proper place down-stream.

A study of the ore-bodies of the Champion Mine, as shown in the maps accompanying Mr. Payne's memoir, and as figured in Major Brooks' report, furnishes additional evidence in support of this theory. The down-stream lenses seem, in some cases, to have been deposited first, as the up-stream lenses lap over them; but the arrangement of the ore is the same in every case, the rich ore at one end and the mixed ore at the other. Some of the lenses consist partly of magnetite and partly of specular. In such case the magnetite is found at the head, the specular next, and the mixed ore at the tail.

In the sketch of the Edwards Mine deposit above, the ore-bodies rest on a thick bed of siliceous ore. The ancient stream

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thus seems to have eroded its bed in this formation, and effected a local concentration of this low-grade ore. It is perhaps unfortunate that its operations were cut short so soon!

The theory of concentration of these ore-bodies by streams receives additional confirmation in the grouping of the lenses in lines or shoots, as at the Dickerson Mine, New Jersey, where some eight or nine lenses have been found in line. In this mine, also, as I am informed by Mr. Moffatt, the mixed ore is always found at the same end of the lenses.

In case the stream were not subject to freshets, we might expect to find a more regular deposit of ore, not divided into lenses, but in one long and regular pod or shoot. Examples of such deposits are not wanting, *e.g.*, the Hurd Mine, and the Ogden Mine, of Morris Co., N. J.

Regular beds, of large lateral extent, would also be formed if the stream be given time enough to gradually shift its bed back and forth across a wide valley, depositing and redepositing the iron sands until finally they become clean and rich. The theory of the original deposition of these beds as bog-ore, however, seems more in accord with the facts observed, and it is interesting to note in this connection that these beds consist usually of specular ore, while magnetite more commonly occurs in lens-formed deposits.

H. S. M.

Michigamme Mine.—The ore of this mine is a fine-grained, blackish magnetite of high grade, averaging sixty-five to seventy per cent. of metallic iron, low in phosphorus and sulphur, and with but traces of other impurities.

Though called a vein, the ore actually occurs in lenticular or pod-like bodies, or the vein is what the miners call "pockety." The general trend of the vein is nearly east and west.

The dip is toward the south forty to sixty degrees. The axis of the individual lenses is inclined toward the west. The lenses at times overlap each other, and at times we find faults, or "crossings," in the body of the lens.

The maximum thickness of the vein—present workings—is about thirty-five feet. The greatest lengths yet observed in any lens were found in the second level, No. 4, where the distance between the east and west crossings was three hundred and ten feet. At this place a little exploration revealed the existence of an overlapping lens, which at the fifth level gave

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a length of over four hundred feet (260 E. + 146 W.), with an average width of about thirty feet.

The hanging wall of these deposits is a quartzite, the foot-wall a magnetic schist, beneath which is a diorite.

The foot-wall is not well defined, as toward the foot the ore becomes mixed and banded with jasper.

The miners, in blasting, try to stop the breaking of the ground at the limit of the clean ore at the foot. It frequently happens that between the true hanging wall (quartzite) and the ore, a slick, or layer of chlorite, occurs. When this is the case and there is a decrease in the dip, *i.e.*, the "hanging bends off," and the chlorite is wet, or "weeping," there is much danger of a fall, and the chlorite has either to be taken down or supported by pillars or stulls. This is the case in the seventh level of No. 4.

This peculiarity in the formation increases cost and danger in the working. In opening up in the next stope of No. 4 (eighth level), it is the intention of the mining captain to take down all the chlorite of the slick, thus sacrificing economy to safety. When no slick occurs, the system of leaving pillars for support is sufficient to render the working safe.

A characteristic of this chlorite is the occurrence in it of so-called black diamonds, *i.e.*, pseudomorphous crystals of chlorite. Some assert these pseudomorphs to be after garnet, others to be after hornblende. It is certain that crystals of *garnet* occur in the rock, and that some of the pseudomorphs on fracture show a shell of chlorite and a kernel of garnet, yet most of them are clearly crystallized in the monoclinic system. These pseudomorphs are much sought after for cabinets.

"Crossings" occur quite frequently. These might be defined to be horizontal faults. We find a number of these between the third and fourth, and fourth and fifth shafts on the lower levels, the throw being generally toward the north, and in some cases as much as thirty feet.

W. F. DOWNS.
E. A. OOTHOUT.

Republic Mine.—The ore-deposits of the Republic Mine are pods or lense-shaped masses of rich specular and magnetic ore of enormous size. The largest body of ore (made up of two or more overlapping lenses?) is over one hundred and sixty feet in thickness and four hundred and thirty feet long,

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and has been mined to a depth of three hundred feet and more, partly by open workings, but more recently by underground cross-stoping, or a species of chamber-workings. The other lenses are from eight to fifty feet in thickness and often four hundred and five hundred feet in length on a level.

The ore-bearing stratum apparently lies in a synclinal trough or basin, the upturned edges of which form a U, surrounding the head of Smith Bay, the outlet of Lake Michigamme, the open end of the U being toward the northwest. The hanging-wall is quartzite and the foot-wall jasper, and in the occurrence of soap-rocks, chloritic schists, etc., the deposits resemble those already described.

Lake Superior Hematite Mine.—The Hematite Mine consists of a vein running due east and west, nine hundred feet in length, and from thirty to one hundred feet in width.

On the north side of the vein is the hanging wall, which is composed of hard jasper rock, and on the south is the foot-wall, composed of a soft schist about six feet in thickness. Behind this is a layer of greenstone. This soft schist is a whitish, putty-like substance, which is so soft that it can be scooped out with the hand. The miners call it paint-rock.

Masses of rock entirely foreign to, and sometimes lying directly in the vein, are frequently found.

The hard ore deposit comes up to the west end of the soft ore vein.

C. L. FITCH.
N. R. WARD.

(To be continued. The next number to contain accounts of the methods of working employed in the different mines.)

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PROFESSIONAL NOTES.

Mining Engineering.

American Institute of Mining Engineers.—Two meetings of this society have been held since the issue of the last number of The Quarterly, one at Stanton, Va., the last week of May, and one lately at Harrisburg, Pa., October 25th to 28th. A large part of the time at the spring and fall meetings of the Institute is devoted to excursions to mines and to metallurgical establishments, and for this purpose the meetings are usually held in or near some mining region. Three such excursions, on special trains furnished by the courtesy of the Pennsylvania, the Cumberland Valley, and the Reading R.R. Companies, formed part of the programme at the last meeting. The excursions, as may be imagined, are both pleasant and profitable. A number of interesting papers were read at the meeting, and many others were announced by title only. Among the latter was a paper by Charles M. Rolker ('75), on the "Chrysolite Mine Fire," and two by John C. F. Randolph ('69), on "The Native Silver Belt of Batopilas, Mexico," and on "A New Form of Stamp Mill."

The Available Bituminous Coal of Pennsylvania.—Dr. Chance, of the Geological Survey, gave an interesting paper, estimating with great care and detail the amount of bituminous coal in Pennsylvania of good quality and contained in beds thick enough for remunerative mining. Excluding poor coal, thin seams, and all coal at considerable depths below water level, or beneath a thick covering of rock, there remains 33,547,200,000 tons of available coal. Assuming that seventy-five per cent. can be obtained in mining, making due allowance for increased consumption, this coal will last about five centuries. According to Dr. Chance, there is enough bituminous coal in Pennsylvania alone to supply the whole world with fuel for eighty or ninety years.

A New Boiler Furnace for the Prevention of Smoke.—Mr. Charles A. Ashburner read a paper describing the Flannery furnace, containing a new device for securing perfect combustion of smoky coals, and thus avoiding the nuisance of smoke, as well as gaining economy of fuel. The coal is burned as usual, on any good form of grate, but the products of combustion, after passing under the boiler, go through a bed of incandescent coke on a supplementary grate at the rear end of the furnace. The effect is the same as in the manufacture of water-gas—the water and carbonic acid are converted into hydrogen and carbonic oxide, which are afterward burned by the admission of heated air in a combustion-chamber, and give off their heat in the return current through the tubes of the boiler. The most important function of the supplementary fire is, however, the heating of the products of combustion again to the point of ignition, and thus securing not only the burning of the smoke, but also the perfect combustion of the carbonic

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oxide that would otherwise be lost. This furnace has recently been tested at Akron, Ohio, burning a very dirty coal slack with most satisfactory results. In eleven days the furnace consumed fourteen tons of coal and 4,200 pounds of coke, or 300 pounds of coke per ton of coal. With coke at $5 and coal slack at $1, the total value of the fuel consumed was $22.60. Without the smoke-consuming device the fuel consumed would have been thirty-five tons for the same length of time, at a cost of $35, showing a saving of forty-eight per cent. of fuel, or thirty per cent. of the cost, by the more perfect combustion effected. A secondary advantage was found in the freedom of the boiler tubes from dust and soot, little or no cleaning being required.

A New Blasting Powder.—Mr. A. C. Rand, of New York, described a new high explosive recently introduced, the invention of Mr. S. R. Divine, known as "Rackarock."* This powder, for greater safety in handling and transportation, is composed of two ingredients, which are combined only when the powder is to be used. Separately, either ingredient is said to be absolutely inexplosive. One of the ingredients is a coarse white powder containing an oxygen-producing salt, and is packed in porous cloth cartridges; the other is a liquid —nitro-benzole. To prepare the powder for use, the cartridges are dipped for a few seconds into the oil, the amount absorbed being regulated by a spring-balance. H. S. M.

Mechanical Engineering.

American Society of Mechanical Engineers.—The annual meeting of this society was held in New York on November 3d and 4th. There were three sessions for the reading and discussion of papers, and an unofficial subscription dinner was held on the last evening. So numerous were the places of professional interest open to the visitors, that it was thought best to organize no formal excursions, and each member followed his own preferences.

The President's inaugural address was a review of past progress in those arts which contribute to the success of the mechanical engineer of to-day. Allusion was made to the malleable alloys of nickel for pump practice, and to a copper alloy of maximum strength. Mr. Charles T. Porter spoke on the classification of drawings in shops by size rather than subject. Numbering and indexing, and concurrent numbering of patterns renders more easy the work of foremen, and diminishes the room required for storage. Professor Robinson gave some of the results of railroad inspection in Ohio as made on behalf of the commissioners for that State. Mr. Root read a paper on the propulsion of screw-vessels by a screw-shaft inclined at sixty degrees to the keel. His experiments were made on a catamaran, and he found the boat to move faster than the theoretical speed due to the pitch of the screw multiplied by its revolutions. The stern of the vessel is lifted, and partly neutralizes the settling observed when the bow rises out of water at high speeds. No economic data were presented. Mr. C. J. H. Woodbury read a paper on fire-protection in mills and

* Dubbed, however, by certain facetious members of the Institute "Rye and Rock," and by others "Divine Wrath."

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factories, and the construction of mill-floors, which was of great value. Two inch wooden doors, covered all over with tin so as to exclude air from the wood, are much better than iron doors between rooms. The wood carbonizes slightly, but no deformation occurs as will happen with iron doors. Isolation of a fire is thus more certain. Fusible metal links, to permit isolating doors to close automatically in case of fire, are approved, and reference was made to automatic sprinklers and fire-pumps. Rotary fire-pumps, driven by grooved friction-gear, are popular. A very excellent type of valve for pipe use was illustrated, which requires no fibrous packing. The stem is non-advancing, and makes a steam-tight ball-joint with a dome in the top of the gland. It has been tested on rock-drill service, which is, perhaps, the most severe of any, and has been found not to "weep" at all. For radiators, and every service where leakage is ruinous and packing is costly, they will meet a great need. Other papers were presented of technical interest, but want of space prevents further allusion. An anti-metric paper, by Mr. Partridge, showed how variable are unsealed measures as found commercially, and called attention to the fundamental relations between the fathom and our other dimensional units, and between the grain and our units of weight. Valuable tables accompanied this paper, which will appear in the Transactions.

F. R. H.

Analytical and Applied Chemistry.

A Universal Flux for Silver Assays.—Geo. L. Stone, '79, gives us a universal flux for the assay of basic silver ores. Its composition is as follows :

Soda	9 parts.
Borax glass	3 parts.
Argol	1 part.

Mix thoroughly, and keep on hand ready for use. For one-third assay ton of ore, fill the crucible about two-thirds full of the flux, adding two or three iron nails when the ore contains much sulphur.

Analysis of Steel.—The members of the American Institute of Mining Engineers are discussing the methods for the determination of the constituents of steels. A paper read at the May meeting (1881), by Wm. Kent, M.E., of Pittsburg, Pa., gave a number of widely differing results for manganese obtained by ten different chemists on portions of the same sample of steel.

From the paper of Mr. Kent and the discussion which followed, it appears that twenty-six different determinations were made, as follows :

Process.	No. of analyses.	Results varied from.
Sodium acetate and bromine	15	0.303 to 1.140
Ammonium acetate and bromine	2	0.351 to 0.380
Nitric acid and potassium chlorate	7	0.313 to 0.390
Not reported	1	0.324

If the single result of 1.14 per cent. by the first-named process be excluded, the results by that process would vary from 0.303 to 0 619. Such a variation with a process used, apparently, by the majority of analysts, naturally attracts attention. Dr. Drown stated that his assistant had found that Swedish filter papers contain small amounts

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of Mn, which may account for some of these discrepancies. Such detailed descriptions of methods as are given in the paper and the discussion suggest also two other possible sources of error: the omission of a second basic acetate separation of the iron, and the direct weighing of the manganese oxide as precipitated by bromine.[1]

Mr. Kent suggests that some method, which in the hands of different analysts would give results accurate within 0.05 per cent., should be adopted. E. W.

At the October meeting (1881) of the same society, the discussion was continued by a paper submitted by Magnus Troilius on the methods for determining C., P., S., Mn., and Si. For all these determinations the methods recommended by Prof. Eggertz are preferred by Mr. Troilius. Besides giving the methods used by himself and other chemists in England and Germany, Mr. Troilius has translated a recent paper by Prof. Eggertz on the colorimetric method of determining the carbon, in which some modifications of the original method are recommended. The paper was printed in advance of the meeting, and analysts in all parts of the country were invited to communicate their methods for comparision and criticism. Some eight or ten chemists took part in the discussion, and papers are expected from others. The material thus collected is to be edited by Dr. Drown, and will, when completed, undoubtedly form a valuable contribution to analytical chemistry.

The main points brought out in the discussion may be summed up in a few words. The requirements of the chemists of steel-works are first for simple, quick-working, and easily applied methods, which at the same time shall be sufficiently accurate for the control of the manufacture of steel; and second for standard methods which shall be perfectly accurate and reliable for use in disputed cases. The color test for *carbon* seems to be almost universally employed, and is considered reliable. For *phosphorus*, the molybdic acid method is employed with subsequent precipitation by magnesia mixture, while for quick work the yellow precipitate is generally weighed, but by some chemists is dissolved, and the ammonia or the molybdenum determined volumetrically. A method of estimating the amount of phosphorus, by measuring the bulk of the yellow precipitate under pressure, was mentioned and excited much interest, but no details were given. For weighing the yellow precipitate, the Gooch filtering crucible—a perforated platinum crucible with asbestos in the bottom—was strongly endorsed by Mr. Dewey, Dr. Dudley, and others. In the determination of *sulphur*, the chief point brought out was that the graphitic residue after treatment with acids was likely to contain sulphur. For *silicon* Dr. Drown's method is generally used. H. S. M.

Sugar Analysis.[2]—J. H. Tucker, '75, has just published a work on sugar analysis—a collection of practical information of value to a

[1] *Vide* Jour. London Chem. Soc., xxxvii., 40, 1880.

[2] A Manual of Sugar Analysis, including the Applications in General of Analytical Methods to the Sugar Industry. With an Introduction on the Chemistry of Cane-Sugar, Dextrose, Levulose, and Milk-Sugar. By J. H. Tucker, Ph.D. 8vo, 353 pp. New York, D. Van Nostrand, 1881.

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sugar analyst, arranged in convenient form for reference. The book is evidently the collection of information which the author has found useful or desirable in the course of his experience in a sugar laboratory, and is therefore eminently practical.

Among the points of information hitherto obtainable only with some difficulty are Clerget's table for the analysis of saccharine substances, Scheibler's table for calculating the percentage of lime carbonate in bone black, the comparative value of the degrees of different optical saccharimeters, the table for the ready calculation of the result of determinations of glucose by the copper reduction test, the explanation of the mode of estimating the probable yield of raw sugars by the method of co-efficients, etc.

Scattered through the book are also the records of a number of interesting experiments made by the author, which are of great practical importance to the analyst, such as the effect of lead salts on the readings of the optical saccharimeter, p. 165; the comparative influence of lead salts and sulphurous acid on the copper-test for glucose, p. 196; the time required for drying the various forms of sugars, syrups, etc., likely to fall into the hands of the analyst, p. 219, etc.

Other features, as the copious references to the literature and a very complete index, contribute to the value of the book as a desirable addition to the library of a sugar chemist or commercial analyst.

E. W.

METALLURGY.

On the Production of Chilling Irons.—The valuable note signed T. E., in the March number of THE QUARTERLY, deserves consideration as treating of an interesting subject from a chemical standpoint. Considered physically, some additional thoughts suggest themselves. In blast-furnace practice we have the usual grades of foundry, forge, mottled and white iron, and we know that any iron which is nearly or quite in the grade of mottled may be chilled or converted into white iron on the surface by rapid cooling. The property of chilling is, however, only of interest with regard to those comparatively few irons which are susceptible of a chill when in a foundry grade. The qualities of the different grades are of the greatest importance, and in the extreme grades almost opposite. Thus, while a foundry iron is soft and may easily be shaped by a tool, white iron resists such treatment. Foundry iron is in one sense strong, will not break under a powerful blow; white iron is brittle. Again, foundry iron is weak, in that it will break under a weight which a white iron will easily endure. For the great majority of castings these contradictory qualities would be eminently injurious; there is known a class of articles in which they are necessary. The most familiar example is that of car-wheels, which in the centre have to withstand continuous jar and on the rim continuous wear.

These considerations may not be new to your readers, but were called to the writer's attention in a series of experiments similar to those described by T. E. Their object was the production of chilling iron for ploughs and car-wheels, and the method essentially that of Bessemer conversion. The originator of the process anticipated deriving remarkable refining qualities from a certain ore which was described as

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containing even gold and platinum, and some elements which defied the detection of the ablest analysts, but which was easily shown to consist of silica 85 per cent., and other not unusual elements. The operation in the converter was devised at first simply to bring the ore and iron into intimate mixture with air as a stirring agent. The result was the ore floated on the surface and was blown out within a minute or two. The experiments were then conducted without the ore and nearly every stage of conversion was tried from merely turning the converter over once, to a period when the product was entirely in the form of mill-cinder. The principal results were obtained, as desired, in cast-iron. Two or three minutes served to convert the iron into white iron, while continued blowing produced a honeycomb iron which sometimes appeared slightly malleable. With the facilities at hand it was impossible regularly to arrest the process before white iron was produced and by far the largest quantity was in that form. It was then suggested that a mixture of this "refined iron" in proper proportion with ordinary foundry pig would produce an iron suitable for car-wheels, and a series of experiments was instituted in mixing. This was done in separate charges in a cupola. Two kinds of test-bars were cast, one to exhibit the chilling property, the other the strength. The strength was tested on an improvised apparatus, which showed the ultimate weight which a bar an inch in section and a foot long, resting on ends with weight in centre, would sustain before breaking. The chilling tests showed generally that the more refined iron used the deeper the chill, but similar mixtures in which refined iron was replaced by ordinary white iron appeared about equally chilled. The strength test showed variable results which accorded with the hardness of the iron. At the same time iron from the blast-furnace was similarly tested, and it was found that ordinary foundry iron stood from 18,000 to 22,000 lbs. and mill-iron from 22,000 to 26,000 lbs. on the scale used. These results did not differ much from iron of similar grade from the refined mixtures, except one test of the latter which showed a result of 34,000 lbs. Presently, however, a test-bar of white iron from the blast-furnace eclipsed everything by requiring over 50,000 lbs.

At this stage it seemed abundantly proved that the inventor had changed his entire method, that the product in the converter could not be made uniform, and that the tests of strength were not the ones suitable for the purpose, and did not show any superiority over ordinary furnace-pig of like grade. Incidentally it was shown that resistance to a load depended primarily on the grade of iron.

It is to be regretted that in the cause of science the experiments were not continued, and, above all, that it was not possible to make chemical tests concurrent with the physical ones. So far as they go, they may serve to explain some of the suggestions in T. E.'s note, and some of the precautions which must be regarded in making future experiments in an interesting field. W. K.

KEMBLE FURNACES, June, 1881.

ORE-DRESSING.

Methods of Separating Minerals of Equal Specific Gravity.—Where the specific gravity of two minerals is nearly or quite the same, the ordinary methods of mechanical dressing fail, and recourse must be had to some other physical or chemical property in which the minerals

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differ enough to make it a practicable means of separation. For example: to separate heavy spar from gray copper, the crushed and sized material has been treated by heat in a suitable furnace, the result of which is to disintegrate the heavy spar by decrepitation, while the gray copper is not affected. On sifting the decrepitated ore the blende passes through the mesh of the sieve which was used to size the ore before heating, while the gray copper remains on the sieve. The results of this method are said to have been satisfactory.[1]

Another similar problem occurs in the separation of blende and pyrite, both minerals having nearly the same gravity, and consequently many middle products will be obtained from the jigs, etc., consisting of mixture of blende and pyrite in various proportions. As a mixture of them has very little commercial value, and a separation by means of water is not practicable, other processes have been invented for this purpose. Thus, magnets have been used for the extraction of the pyrites, after it has been converted into magnetic iron pyrites by roasting of the ore mixture with an addition of carbonaceous substances.

Another process for the separation of blende and pyrite, which is still more simple and efficient, was lately devised by director F. Büttgenbach, of the Lintorf Lead Mines, near Düsseldorf, and is described in *Engineering* for Sept. 9, 1881, as follows:

The very productive lodes of these mines yield a mixture of lead ore, galena, iron pyrites, and blende, accompanied by quartz, limestone, and bituminous shale. Upon the very perfect machinery of the Lintorf ore-dressing mill all other substances are thoroughly separated from each other, with the exception of certain sizes of pyrites and blende, which are too small for hand-picking. As the specific weight cannot be utilized for the separation of these, Mr. Büttgenbach has hit upon the excellent idea to make the differences of cohesion in the two minerals available for their separation; for blende is much softer, and can be much easier crushed than pyrites. For the purpose Mr. Büttgenbach employs the Vapart centrifugal disintegrator, the reducing action of which can be so nicely regulated that the blende can be reduced to sand and the pyrites left intact. The first trials with a Vapart disintegrator No. 2, showed that with 600 revolutions a minute the pyrites was too much broken up; the same happened with 450, but with 225 too much blende remained unbroken, while with 325 revolutions the blende ore was almost perfectly reduced to fine sand, and the grains of pyrites, 10 mm. or 6 mm. in size, kept their volume, so that both could be easily separated by simply passing them over a sieve. This simple operation is a great commercial success, as it enhances the value of a substance which was almost valueless because it was too poor for an ore of zinc, and too impure for the burning kilns of chemical works. The blende sand, which contains from 50 to 55 per cent. of zinc, is now worth about £4 a ton, and the grains of pyrites separated from it sell easily at 12s. a ton. The process answers equally well for all sizes, from 25 mm. to 4 mm., and one Vapart apparatus is capable of separating from 2½ to 3 tons an hour.

H. S. M.

[1] We learn from Mr. H. V. Furman ('81), who has just returned from the West, that this method is now in practical use in the San Juan region of Colorado.

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SCHOOL OF MINES NOTES.

THE recommendations of the committee appointed to consider the expediency of increasing the requirements for admission to the School of Mines have been adopted, and go into effect next spring. The report of the committee was printed in our last number.

A NEW course, that of Architecture, has been added to the five already existing in the School, and Prof. W. R. Ware, formerly of the Massachusetts Institute of Technology, of Boston, now lectures twice a week to the second year students pursuing that course. The arrangements for the third and fourth years have not yet been announced. An interesting memorandum on the scope of the proposed course, by the Professor, will be found in another place.

WE are pleased to note that Mr. J. K. Rees, A.M., E.M., who graduated from the School in 1875, and became Professor of Mathematics and Astronomy in Washington University, St. Louis, has returned to the College as Instructor in Geodesy and Director of the Observatory. He will deliver one lecture on geodesy per week to the fourth-year engineering students, and two lectures per week on astronomy to the third-year students. Newcomb and Holden's Astronomy is used as a text-book. A summer class in geodesy will also be organized for the civil engineers.

THE first meeting of the season of the Chemical Society was held Thursday, October 20. President Wittmack called the meeting to order, Mr. Wiechmann read his "notes," in which he proposed a terminology for minerals, by means of which the system of crystallization of each is made apparent by its name. After a discussion of the merits of the plan by several members, the Society adjourned.

A special meeting of the same Society was held on Friday, October 28, 1881, at 3 o'clock, P.M., President Wittmack in the chair. A motion to change the time of the meetings to every second and fourth Thursday afternoon, at 4 P.M., was carried. After the election of Mr. F. W. Traphagen, '82, to the position of Secretary, made vacant by the resignation of Mr. J. Lynch, the meeting adjourned.

THE Registrar's books show that up to date (November 12th) there are 266 students in the School, 41 in the fourth class, 47 in the third class, 83 in the second class, 94 in the first class, and one post-graduate. The School showed a steady growth in the four regular classes, curiously enough, of exactly ten men per year up to last year, the totals in successive years since 1878 being 222, 232, and 242.

ON account of the dropping of special students last year, and by reason of the recent legislation with regard to the payment of post-graduate fees, the total number of students in the School of all classes, has remained about stationary for three years. The growth, however, is none the less real. The proportion of students selecting the

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different courses remains about the same as in previous years: sixty-eight per cent. taking Mining Engineering, twelve per cent. Civil Engineering, and twenty per cent. the course in Analytical and Applied Chemistry.

The School is thus, not without reason, called a School of Mines, and in point of numbers, at least, is the leading mining school of the country. From the last report of the Bureau of Education, we find that thirty-one men took the degree of Mining Engineer in 1879. Of these, twenty were from the School of Mines, and of the others, four received their degree at the Washington University of St. Louis, under the instruction of William B. Potter, E.M. (S. of M., 1869), Professor of Mining and Metallurgy, and one graduated from the University of Wisconsin, where R. D. Irving, E.M., Ph.D. (S. of M., 1869) is Professor of Geology and Mineralogy.

WORK on the new library building has been considerably delayed by the peculiar nature of the ground upon which the east wall was to have been built. At this end of the site, and directly on the line of the proposed foundation, the rock, which in other places is very near the surface, suddenly disappears, dipping downward at an angle of about seventy-five degrees with the horizon, and giving place to a very soft, muddy soil. A deep excavation was made in this soil along the face of the rock, but as no change was observed, either in the dip of the rock or the treacherous nature of the soil, the work was abandoned and the excavation filled in. It has now been decided to alter the original plan of the building, running it further west by an amount about equal to the required shortening at the east end. To do this it has been found necessary to tear down and rebuild a considerable amount of the masonry already in place, whence the delay.

THE question of post-graduate fees has during the last four weeks been a subject of much discussion among recent graduates and students who are intending to take a post-graduate course in the school. A petition was submitted by those interested to the Board of Trustees at their last meeting, and referred by them to the Committee on the School of Mines. The petition asks for the remission of the annual fee of $150, imposed this year for the first time, on graduates returning to the school. The main arguments advanced in the petition in favor of this request are that the student should be encouraged to devote himself to original scientific investigation, which will reflect credit on himself and on the school, the course pursued by similar technical institutions in this country and abroad being instanced, many of which even afford pecuniary assistance to those undertaking such work. It was further argued that the graduate undertaking such investigations not only has little opportunity to make money by professional work, but is almost invariably, and of necessity, put to considerable personal expenditure. The argument was also advanced that there is no special course of lectures or of instruction for post-graduate students, and that, from the nature of their work, they make but little demand on the time or attention of the professors and instructors, and, finally, that those undertaking such post-graduate work are always few in number, and are not allowed to occupy room to the disadvantage of regular students.

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ANNUAL REPORT OF THE PRESIDENT.—The annual report of President Barnard, dated June 6, and recently printed by the College, is a most valuable document, containing able and interesting discussions of many questions of vital importance to the College and its associated schools, *e.g.*, the practicability of introducing a system of self-government by the students, instead of burdening the faculty with questions of discipline; the best means of removing or overcoming existing imperfections in the preparatory schools, and their systems of instruction; the proposed re-establishment of the Columbia Grammar School; the admission of women to Columbia College; the proposed establishment of a new department of Educational Science, with lectures on the history, theory, and practice of education, etc.

We extract from the report that portion devoted to the work of the School of Mines as of interest to students and graduates.

SCHOOL OF MINES.—In this School the resolutions of the trustees providing that in future only such applicants for admission shall be received as propose to become candidates for degrees in some one of the regular courses of study, have been in operation for the first time. Their effect has been, naturally, to reduce to some extent the total number of names upon the list, but it has been attended in other respects with obvious advantages. The results of the year, as shown by the record of scholarship, have never been more satisfactory. In the reports of the summer classes in mining and mechanical engineering, conducted, the first by Professor H. S. Munroe, and the second by Instructor F. R. Hutton, will be found evidence of the practical usefulness of these exercises, and of the continued interest felt in them by instructors and students. The summer class in mining have been greatly indebted to the kindness of the Musconetcong Iron Co., of New Jersey, and the Westmoreland Coal Co., of Pennsylvania, for the courteous manner in which the mines of those companies were thrown open to the class, and for the facilities afforded them in pursuing their practical studies. The class in mechanical engineering are under similar obligations to the proprietors of the Quintard, the Morgan and the Delamater Iron Works of this city, for the free range of their works allowed them, and other courtesies.

The School of Mines, since it was first opened to the public in 1864, has undergone important modifications of plan, and has largely extended the comprehensiveness of its scheme of instruction. It was originally designed to be, and during the first few years of its existence it was, merely a school for educating men to scientific methods of extracting from the earth its mineral treasures. To the course of instruction in mining engineering were added, later, other courses in metallurgy, in civil engineering, in analytical chemistry, and in geology and palæontology. The School has, therefore, long since ceased to be simply a School of Mines, and it might more properly be styled a School of Applied Science. Recent resolutions of the Trustees have added another to the courses of instruction above enumerated, which is perhaps of hardly less importance than any one of those, by creating a Chair of Architecture, and electing a professor of eminence to fill it, who is expected to enter upon duty at the opening of the ensuing session. It is gratifying to be able to state that this effort to enlarge the usefulness of our School, in a department of education in which in our country very little has as yet been systematically attempted, has

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awakened a lively interest among the professional architects of our city, some of the most eminent who have kindly favored us with their council and advice. In no respect has the advancing culture of our age manifested itself in recent years more strikingly than in the rapid growth of a taste for the beautiful in constructive art. This makes itself felt, not only in regard to large and costly public edifices, but also almost equally as to the most modest private dwellings. Along with the growing demand for beauty, there is also an increasing demand for such skill in constructions as shall secure in dwellings the greatest convenience of arrangement, together with such conditions as to heating, ventilation, and drainage as are most favorable to health. The profession of the architect has, therefore, risen into an importance which makes it extremely attractive to young men, especially to such as have native tastes for art. Most of the gentlemen who adorn this profession in our country, and who are not self-made architects, have found abroad advantages for instruction which hitherto our country has not afforded. The only school of architecture which has heretofore been opened in the United States has been that which was established about ten years ago by our own Professor-elect. Our Professor comes to us, therefore, with the advantage of a double experience—experience as a teacher and experience as a professional man. His practice as an architect has enabled him to form very distinct notions of what it is necessary that a man in his profession should know, and his experience as a teacher has afforded him opportunity to ascertain the most efficacious methods of imparting that knowledge. The soundness of the views of the Professor-elect as to the character of the course of instruction proper to be pursued in our school, may be judged from a brief discussion, in the form of a memorandum on the subject appended to this report,[1] which was drawn up by him at the request of the undersigned, and in which the claims of the several elements which should enter into the education of an architect are so sensibly weighed and so justly balanced as to leave no question that the writer is master of his subject.

Another important enlargement of the scheme of instruction in the school has been accomplished by the appointment of an instructor in geodesy, who is also to be the director of the observatory about to be constructed to replace the inferior structure of that character recently demolished. This is an improvement which has been long regarded as desirable, and which was brought to the attention of the Trustees, as such, in the report of the undersigned for 1879. The education of a civil engineer can hardly be regarded as complete unless he is acquainted with the astronomical methods necessary for the determination of the exact relative position of points on the earth's surface. It is, of course, possible to acquire a theoretic knowledge of these methods from books; but it is only actual practice that can make them available for any useful purpose. The resolution authorizing the appointment of an instructor in geodesy, therefore, very judiciously provides that the officer so appointed shall give practical instruction in the field during the summer in geodetic methods. There is every probability that arrangements may be made with the Superintendent of the Coast and Geodetic Survey of the United States, and with the

[1] Reprinted (pp. 1–4) in this number of THE QUARTERLY.

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Director of the State Survey of the State of New York, whereby the members of our geodetic class may be permitted to observe and study the practical operations in progress in the conduct of those great works, and may, perhaps, be allowed to participate in working out their details.

With the completion of the new observatory it will also become possible for our College to take its part in the regulation of the public time—a matter of great and general interest to the community, in which the leading observatories in other parts of the country are already actively engaged. . . . Our observatory will have the advantage of occupying one of the most favorable positions for participating in this work; and the Director-elect will bring to it an experience acquired in connection with similar operations in which he has been interested in the West. It cannot but be highly advantageous to the College thus to make itself actively instrumental in the promotion of an object of so great public utility. . . .

F. A. P. BARNARD,
President.

COLUMBIA COLLEGE, June 4, 1881.

BOOKS ADDED TO THE SCHOOL OF MINES LIBRARY SINCE MAY, 1881.

ENGINEERING AND MATHEMATICS.

Bartlett, W. H. C. Elements of Analytical Mechanics. 8vo, 530 pp. New York.

Courtney, Edward H. A Treatise on the Differential and Integral Calculus. 8vo, 501 pp. New York, 1876.

Fairly, W. Theory and Practice of Ventilating Coal Mines. 8vo, 72 pp. London, 1879.

Purdy, William. Colliery Explosions and Safety Lamps. 8vo, 55 pp. London, 1880.

Randolph, John C. F. Report on the Pastrana Mine, Batopilas, Mexico. 4to, 32 pp. New York, 1881.

Swinney, Alfred J. G. Handbook to Examinations for Colliery Managers. 8vo, 128 pp. London, 1878.

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Van Wagenen, T. F. Manual of Hydraulic Mining. 8vo, 93 pp. New York, 1880.

Wardle, W. Reference Book of Practical and Scientific Information for the Use of Colliery Managers. 8vo, 191 pp. London, 1880.

Webber, Samuel. Manual of Power for Machines, Shafts, and Belts. 8vo, 105 pp. New York, 1879.

Wheeler, J. B. Elementary Course of Civil Engineering. 8vo, 472 pp. New York, 1877.

Wood-working Tools: How to use them. 12mo, 102 pp. Boston, 1880.

GEOLOGY, MINERALOGY, AND NATURAL HISTORY.

Britton, N. L. A Preliminary Catalogue of the Flora of New Jersey. 8vo, 233 pp. New Brunswick, 1881.

——— The Geology of Richmond County, New York. 8vo, 21 pp. New York, 1881.

Cossham, Handel. Faults in the British Coal Field, and how Caused. Pamphlet, 80 pp. London, 1879.

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Marsh, O. C. Odontornithes. Vol. VII. U. S. Exploration of the 40th Parallel, 4to, 291 pp. Washington, 1880.
Parton, T. Notes on the Shropshire Coal Field. 8vo, 9 pp. London, 1868.
Wallace, Alfred Russell. Island Life. 8vo, 522 pp. New York, 1881.
Weisbach, Dr. Albin. Charakteristik der Classen, Ordnungen und Familien des Mineralreiches. 8vo, 57 pp. Freiberg, 1880.
Whitney, J. D. The Auriferous Gravels of the Sierra Nevada of California. 4to, 569 pp. University Press, 1880.
Wilson, Daniel. Prehistoric Man. 2 vols. 8vo, 800 pp. London, 1876.
Wolf, Heinrich. Geologische Gruben-Rivier-Karte des Kohlen beckens von Teplitz-Dux-Brux im Nordwestlichen Böhmen. 8vo, 19 pp., folio 15 Blätter. Wien, 1880.
Woodward, Henry. The Dawn and Development of Life on the Earth. Pamphlet.
Zittel, Dr. Karl. Ueber den Geolischen Bau der lilyschen Wüste. 8vo, 47 pp. München, 1880.

CHEMISTRY.

Austen, P. T. Dinitroparadibrombenzols and their Derivatives. 8vo, 5 pp. New York, 1881.
Kopp, Herman. Geschichte der Chemie. 4 vols. 8vo. Braunschweig, 1843.
The People vs. Daniel Schrumpf. Misdemeanor. Adulteration of Milk. 8vo, 283 pp. New York, 1881.
Winkler, Dr. Clemens. Anleitung zur Chemischen Untersuchung der industrie Gase. 8vo, 210 pp. Freiberg, 1877.
Wurtz, Ad. The Atomic Theory. 8vo, 344 pp. New York, 1881.

JOURNALS, REPORTS, AND TRANSACTIONS (NEW).

Anthropological Society of Washington. Abstract of Transactions for 1880 and 1881. 8vo, 150 pp. Washington, 1881.
Association pour Prévenir les Accidents de Machines. Compte Rendus. 1867 to 1876. 8vo. Mulhouse.
Bureau of Education, Bulletins of:
Comparative Statistics of Elementary Education in Fifty Principal Countries. Chart.
Clark, F. W. Report on the Teaching of Chemistry in the United States. 8vo, 219 pp. Washington, 1880.
Construction of Library Buildings. 8vo, 26 pp. Washington, 1880.
Fifty Years of Freedom in Belgium. 8vo, 8 pp. Washington, 1881. Bureau of Education.
Library Aids. 8vo, 10 pp. Washington, 1881.
Powell, J. W. Introduction to the Study of the Indian Languages. 4to, 288 pp. Washington, 1880.
Relation of Education to Industry. 8vo, 22 pp. Washington.
Spelling Reform. 8vo, 36 pp. Washington, 1880.
Engineers' Club of Philadelphia. Proceedings. Vol. 2.
Kerl, Bruno. Repertorium d. Technischen Journal-Literatur. Jahrgang 1879. 8vo, 499 pp. Berlin, 1881.
London Inst. of Civil Engineers. Charter, By-Laws and Regulations and List of Members. 8vo. London, 1881.
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——— Report of Her Majesty's Commissioners. 2 vols. 8vo. London, 1880.
South Carolina. First Annual Report of the Commissioner of Agriculture. 8vo, 115 pp. Charleston, S. C.
South of Wales Institute of Engineers. Proceedings. Vols. 1 to 12. 1857-1881. 12 vols. 8vo. London.
Western Society of Engineers. Proceedings, 1880. 8vo, 194 pp. Chicago, 1880.

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GRADUATE DEPARTMENT.

HENRY GILBERT CLARK.

On August 2d, 1881, Henry Gilbert Clark, E.M., C.E., class of '77, died of typhoid fever at Aurora, Nevada. His illness began during the early part of the summer, and at one time he was thought to be rapidly convalescing, but a sudden relapse proved too much for his already weakened constitution, and death quickly ensued.

Cut off at the early age of twenty-five, while still fresh in the memories of his class-mates, he will be remembered as one who, by his course at the School, made a host of friends. His class-mates esteemed him one of their ablest and strongest members. A faithful student and hard worker, with an energetic will and genial manners, he was regarded as one who would succeed in his profession. His career since graduation but served to confirm that opinion. The position of chemist to the Bergen Port Zinc Works was filled by him with credit for two years. Wearying of the somewhat limited field in the East, he began to report on mining properties in the West while still acting as expert for his former employers, and at his death held the responsible position of Superintendent to the Cortez Mining and Milling Company, at Aurora, Nevada. Another name must be added to our list of the dead, and another familiar face will be missed at our annual reunions.

C. L. C.

The following list of graduates, arranged according to locality, is interesting as showing how widely scattered are the School of Mines' men. It has been compiled with two objects—first to inform those who are travelling about this country, and over the world, where they may expect to meet fellow-graduates, and second to inform the older graduates of the whereabouts of the younger men in their vicinity (and *vice versa*) that they may be enabled to make each others' acquaintance. For exact address in each case the reader is referred to the Graduate Lists published in the March and May numbers of last year, and to the corrected lists which appear from time to time in The Quarterly. The long list of men in New York City might perhaps have been divided into three classes: 1st, those permanently located in the city or vicinity; 2d, those who make New York a headquarters only—a large proportion; and, 3d, those who are unemployed—fortunately a few men only.

Cuba.—Brinckerhoff, '78; Francke, '80.
Brazil.—Gordon, '71; Barros, '77; Jordáo, '77.
England.—Lyman, '78.

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GERMANY.—Kunhardt, '80; Navarro, '80; Hemmer, '81; Neyman, '81.
JAPAN.—Hasegawa, '78; Matsui, '78; Nambu, '78.
MEXICO.—*State of Chihuahua:* Holbrook, '76; Adams, '78; Leggett, '79; Sheldon, '79; Browning, '80. *Sinalo and Durango R. R.:* Griswold, '81; Roberts, '81; Swain, '81; Wilson, '81.
PERU.—Pazos, '78.

ALABAMA.—Noble, '79.
ARIZONA.—*Globe:* Devereaux, '78. *Prescott:* Rutherford, '79. *Tombstone:* Church, 67; Van Blarcom, '76. *Tucson:* Hunt, '76; Milliken, '79.
CALIFORNIA.—*San Francisco:* Adams, '67; Lilienthal, '70; Leavens, '75; Downing, '78. *Bodie:* Noyes, '75; Harker, 79.
COLORADO.—*Colorado Springs:* Parsons, '68; Strieby, '78. *Georgetown:* Tilden, '76. *Golden:* Ihlseng, '75. *Gunnison:* Craven '76; Cornwall, '76; Reed, '77; Cornwall, '79. *Kokomo:* Cloud, '79. *Leadville:* Bruckman, '69; Van Wagenen, '70; Williams, '74; Iles, '75; Rolker, '75; Bruen, '76; Helleberg, '77; Nichols, '77; Booraem, '78; Fernekes, '78; Koch, '79; Clark, '80. *Ouray,* Mathis, '79. *Rico:* Garlichs, 80. *Silver Cliff:* Parker, 80.
DAKOTA.—*Deadwood:* Jenney, '69. *Esmeralda:* Buckley, '77; Murphy, '77.
DISTRICT OF COLUMBIA.—*Washington:* Floyd, '77.
GEORGIA.—*Atlanta:* Chazal, '81.
IDAHO.—*Rocky Bar:* Rogers, '77.
ILLINOIS.—*Aurora:* McCulloh, '78. *Bloomington:* Tuttle, '67. *Chicago:* Giddings, '67. *Joliet:* Meissner, '80.
LOUISIANA.—*New Orleans:* Stewart, '75; Eastwick, '79.
MASSACHUSETTS.—*Boston:* Schack, '68.
MICHIGAN.—*Ann Harbor:* Joy, '75. *Delaware Mine P. O.,* Klepetko, '80. *Marquette:* Wetmore, '75.
MISSOURI.—*St. Louis:* Robertston, '68; Potter, '69; Riggs, '71; Stone, '79.
MONTANA.—*Helena:* Nettre, '69. *Wickes:* Stearns, '81.
NEBRASKA.—*Omaha:* Hoyt, '76.
NEVADA.—*Dayton:* Wheeler, '68. *Eureka:* Radford, '77. *Egan Cañon:* O'Connor, '81. *Winnemucca:* Van Lennep, 67.
NEW JERSEY.—*Bethlehem:* Randolph, '76. *Dover:* W. A. Smith, '68; Canfield, '73. *Newark:* Pool, '75; Karr, '78; Murphy, '78; Starr, '79. *New Brunswick:* Austen, '72. *Orange:* Lindsley, '70. *Princeton:* Cornwall, '67; Marsh, '79. *Stanhope:* Moffatt, '68.
NEW MEXICO.—*Georgetown:* Smalley, '80. *Silver City:* Ross, '76. *Tombstone:* Judd, '81.
NEW YORK.—*Brewsters:* Ruttman, '80. *Buffalo:* Douglas, '81. *Cornwall on the Hudson:* Ledoux, '81. *Furnaceville:* Parrott, '70. *Hamilton College:* Chester, '68. *Potsdam:* Heath, '67. *Staten Island:* Wardlaw, '76. *West Point:* M. Smith, '80.
NEW YORK CITY AND BROOKLYN.—
'67—Bridgham, Brown, Brownson, Gracie, Hale, Harmer.
'68—Van Arsdale, Barnard, Baxter, Carson, Geer, Pistor, Platt, Schermerhorn, Stallknecht.
'69—Hooker, Munroe, Randolph.
'70—Haight, Ingersoll, Waller.
'71—Fales, Goldschmidt, Ricketts, Roberts.
'72—McDowell, Sloane, Wendt.
'73—Colton, Mott, Webb, Williams.

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'74—Allen, Benedict, Cameron, Murray, Olcott, Rees.

'75—Jackson, Putnam, Russell, Thompson, Wells, Tucker.

'76—Garrison, Gratacap, Hall, Hamilton, Hutton, King, Love, Maghee, Morewood, O'Grady, Wallbridge.

'77—Beard, Behr, Canfield, Caldwell, Colby, Constant, Cornell, Hildreth, Ihlseng, Mackintosh, Norris, Sage, Smeaton, Smythe, Thacher (in California?) Van Boskerck, Waterbury.

'78—Cushman, Butler, Drummond, Elliott, Haas, Hodges, Holden, Johnson, E.M., Johnson, G.H., Morewood, Morris, Munsell, Newberry, Noyes, Owen, Parker.

'79—Britton, DeLuze, Haffen, Hathaway, Hollerith, Hollick, Johnson, Johnston, Mayer, Merwin, Munroe, Neftel, Nesmith, Reed, Suydam, Williams.

'80—Beebe, Benjamin, Butler, Churchill, Engel, Greene, Greenleaf, Hallock, Hendricks, Hooper, Hopke, Hudson, Mattison, Merritt, Parks, Robinson, Torrey, Tonnélé, Walker.

'81—Andresen, Aschman, Braschi, Bush, Colby, Dunham, Elliott, Furman, Leary, Little, Meserole, Pitkin, Raymer, Richmond, Share, Starr, Van Sinderen, Vulté, Wiechmann, Williams.

NORTH CAROLINA.—*Chapel Hill:* Brugman, '80. *Charlotte:* Hanna, '68. *High Point:* Sawyer, '81. *Salisbury:* Cady, '77.

OHIO.—*Cincinnati:* Lord, '76. *Cleveland:* Foote, '76. *Oberlin:* Wright, '75. *Toledo:* Tuttle, '81.

PENNSYLVANIA.—*Philadelphia:* Lillie, '74; Ludlow, '79. *Pittsburg:* Lenox Smith, '68; Robertson, '71; Palmer, '78; Singer, G., '80; Singer, G. H., '80. *Riddlesburg*. Kelly, '77.

RHODE ISLAND.—*Newport:* Willis, '78.

TENNESSEE.—*Nashville:* Campbell, '69. *Tracy City:* Davis, '78.

TEXAS.—*Harrisburg:* Hollis, '78.

UTAH.—*Ogden:* Bleecker, '81. *Park City:* Rhodes, '79. *Salt Lake City:* Knapp, '70; Terhune, '70; Rhodes, '74; Schneider, '74; Wheeler, '80.

WISCONSIN.—*Madison:* Irving, '69.

WYOMING.—*Cheyenne:* Olmstead, '78. *Rawlins:* Blydenburgh, '78.

DECEASED.—MacMartin, '68; Blossom, '69; Newton, '69; Jenney, '72, Lamson, '75; Clark, '77; Priest, '77; Stewart, '79.

ADDRESS UNKNOWN.—Coursen, '68; Pfister, '75; Perry, '78 (Colorado?).

R. D. IRVING ('69) promises, during the present year, to contribute to THE QUARTERLY. Professor Irving's field-work and writings on the Geological Survey of Wisconsin, the latter forming a large part of the two bulky volumes published by the Survey, and his numerous articles and papers, published in journals and the transactions of scientific societies, speak for themselves. Any article from his pen is certain to be both interesting and of great value to the readers of THE QUARTERLY.

A. F. WENDT, E.M. ('72), we see from a card in the daily papers, started recently for Tombstone, Arizona, on professional business. We send by him our regards to Church and Van Blarcom.

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DR. J. H. TUCKER ('75) is located at 120 Front Street, New York, as analytical chemist, giving special attention to sugar work. Mr. Tucker's long experience in connection with sugar refineries in Baltimore and New York, and researches on sugar, have recently been embodied in a valuable work, published by Van Nostrand, a review of which will be found in another place.

D. A. JOY ('75) is located at Ann Arbor, Michigan, and must now be ranked among our medical graduates, as we see that he has taken the degree of M.D. in the University of Michigan.

F. N. HOLBROOK ('76) writes from Corralitos, Chihuahua, Mexico, where he is Superintendent of the Corralitos Mining Co., as follows: "There are three of us School of Mines' men here, and to talk about 'old times' in the school, and especially in the 'Assay Lab.,' is a recreation we often indulge in after our hard day's work. We are busy, as you can imagine, erecting furnaces and getting things into shape, which for the past six weeks has been vexatious work, on account of the continual rains that destroy in a few hours the work of days. While the work goes on here, I am out in the mountains with the diamond-drill, prospecting.

"To-day, being mail day, I came in for letters, etc., but, as usual, the mail-rider has not come, and it is doubtful whether he will be here this week. What with the Indians and swollen rivers, we get our mail when we can. This is a most beautiful country at this season of the year. Everything is green, the grass growing to the very summits of the mountains. There is one great drawback, and that is the Indians. We work, while in the mountains, with revolver and cartridge-belt strapped about the waist, and, most of the time, rifle over the shoulder or within grasp. The red devils came on to our grant again yesterday, on the western limit, and at what moment they will be at the other end where we are working '*quien sabe.*' "

S. A. REED ('77), of Irwin, Gunnison Co., Colorado, promises us some contributions on the subjects of ore-dressing and ore-sampling. He says: "We intend putting up concentration works for rather difficult ores, and may have something interesting on that subject."

WILLIAM KELLY ('77) writes from Kemble Furnaces, Pennsylvania, sending us results of a series of experiments on the production of chilling irons, which we publish in this number, as they were received too late for the spring number. Mr. Kelly holds the position of Superintendent to the Kemble Coal and Iron Co., Riddlesburg, Bedford Co., Penn.

F. W. FLOYD ('77) writes us from the United States Engineer Office, the headquarters of the Geographical Surveys West of the 100th Meridian, Washington, D. C., as follows:

"I have just received your letter accompanying THE QUARTERLY, and take pleasure in forwarding my subscription, with the hope that the effort to establish a communicating medium between the graduates will prove a success.

"As you perhaps know, the Geographical Survey is now completing its reports and maps; all field-work having been suspended,

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and I am compiling the results of the geodetic and hypsometric computations.

"Probably the most distinctive feature of the field-work is the determination of the vertical element, or altitude, by means of an aneroid profile of the meandered route for each day, usually a distance of from fifteen to thirty miles. Nearly all railroad engineers find the aneroid barometer useful on preliminary lines, and our work shows that when checked with a mercurial barometer, it can be applied to a greater extent than seems possible with such a variable instrument."

CORTLANDT E. PALMER ('78), writing from the office of the Engineering Department of the Pennsylvania Company, Pittsburgh, Pa., promises us an article on "Plate-Girder Bridges," as soon as he can find time for it. He says, however: "I have been too busy to think of anything but 'railroad' since I returned from my New York trip; holidays, Sundays, and every other day crowded with work, and half the nights beside. I have just completed the location of the branch road of which I spoke to you, and shall have charge of the construction, which begins at once." On the official letter-heading we see that the Pennsylvania Company are operating the Pittsburgh, Fort Wayne & Chicago R. R., and branches, the C. & P. R. R., the A. & P. Ry., the E. & P. R. R., and the N. W. O. Ry. From this formidable array of initial letters, we infer that Mr. Palmer's chances for leisure are small.

We understand that our rifle-team graduate, Mr. C. E. Blydenburgh ('78), has located permanently in Rawlins, Carbon Co., Wyoming Territory, where he is actively engaged in engineering and assaying work, besides holding the position of Territorial School Inspector.

T. H. LEGGETT ('79) writes from the Soledad Mine, near Batopilas, Chihuahua, Mexico, under date of August 31, 1881, as follows:

"My trip down was a very cold one, especially in Colorado, but I succeeded there in seeing the Bobtail and Freeland Mines pretty thoroughly, not accomplishing much in Georgetown, however, on account of Christmas intervening. It was the middle of February before I finally reached Batopilas, having to wait a month in Chihuahua for mules on which to cross the mountains, but while there I spent some very pleasant days with Browning, of '80, going with him to visit the mines at Cusihuiriachic, where he now is, and also visiting the Santa Eulalia Mines of the same company, which are just out of Chihuahua.

"As to the work I have been engaged in: For the first six weeks I was very busy surveying the catacombs and labyrinths known as the Mines of San Miguel—Mexican workings so confined and irregular that I used chiefly the string and swinging compass.

"After finishing my maps, I went up on the mountains a couple of miles from the hacienda, to take charge of the Camuchin S. M. Company's property, where I was engaged in running two tunnels, and sinking two shafts ultimately to connect with them, first having to trace one vein as far as possible and direct tunnel to cut it by survey, which I succeeded in doing, and in Camuchin to explore the old works, finally locating and partially cribbing a new shaft as a working point.

"After getting things a little more ship-shape there, though much

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remains to be done, I came here—a day's ride south of Batopilas—in July to examine and exploit three veins, hitherto sufficiently scratched by the Mexicans to show they contain some metal.

"The country is entirely different from that of the native silver-bearing districts near it, being entirely volcanic rock, the veins segregations of heavy spar, carrying gray copper, galena, and sometimes black silver.

"As I have only a force of about twenty men, shall have to be here rather longer than I anticipated for this prospecting work. Mexican cheap labor I find thus far pretty well offset by its scarceness and unreliability. Very few of the men are good miners, in the simplest sense of the word, that of pointing holes and sharpening drills; each set does its own smithing here.

"As have had to do both in breaking in some of the green 'rancheros' around here, have found the practical work in that line at Mineville very useful.

"At present am running drifts on two of the veins, and sinking a shaft on the third. Busy also in sampling for assay, making sketch maps for reports, etc. As veins are all far apart, spend considerable time on mule-back; but mining life in this country, in its roughest, is not so bad, I judge, as they have it in our Western mountains.

"Am living at present in a 'jacal,' or palmetto-thatched hut, on the mountain side, with cows at hand to furnish milk for coffee, finding corn-cakes, or 'tortillas,' and fresh cheese a fair substitute for bread and butter, and sweet corn and beans at hand in the ranches.

"Shall return in a month or so to the hacienda, where Mr. Walter M. Brodie, of Clausthal, is in charge of the metallurgical work, with W. J. Adams, class of '78 assisting him.

"Received the March copy of THE QUARTERLY, and was so much pleased with it and its object of holding together the graduates, that I not only sent a subscription—received, I hope, ere this—but started a few notes on harbor surveys, which lie at present unfinished, from press of work. If, however, you think the subject acceptable, will endeavor to finish them—I pause to brush off a too-familiar centipede crawling over the table. They, with the scorpions and tarantulas, form a very unpleasant feature of the country, but, like all other things, one has only to become accustomed to them."

F. D. BROWNING ('80), writing from Cusihuiriachic, Chihuahua, Mexico, says that the work of "looking after the reduction of our ores, the mining operations, and the construction of new works," keeps him busy. "When I first came here I found the tailings from lixiviation running twenty ounces per ton, with probably an equal loss through the chimney. I kept my assay furnace hot most of the time, till I reduced the loss in tailings to about five ounces, and had dust chambers built to reduce the chimney losses. By deferring the addition of salt till near the end of the roasting, instead of adding it at the beginning, with the ore, as had been the custom, the most of the loss of silver was prevented.

A. D. CHURCHILL ('80) holds this year the position of Assistant in Drawing to Mr. Pistor. With the increase in the size of the classes which has taken place this year, without a corresponding expansion

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of the walls of the drawing-room, Messrs. Pistor and Churchill find much difficulty in placing their men.

A. M. PARKER ('80) writes from Silver Cliff, Colorado: "I was examining, a short time ago, some Cornish jigs at a custom concentrating mill near by. They were endeavoring to concentrate ores of nickel and cobalt. The specific gravity of the gangue and ore was very nearly the same, and consequently the success was not over encouraging. The concentrating mills here are in some cases well designed, but at times run rather loosely, with large losses."

P. E. CHAZAL ('81) is at work in Northern Georgia, examining mineral properties and prospecting in the interest of some parties in Atlanta.

A. L. BEEBE ('80), is Assistant in Assaying, and may now be daily seen in the Assay Laboratory, where he is doing excellent work, having thoroughly fitted himself for the position during the past year.

CHAS. P. BLEECKER ('81) is located as assayer at Ogden, Utah, where he is engaged in thoroughly testing the deposits of that district. He draws attention to an interesting case of a magnetic sand, which, while showing well in the pan, for some reason does not give good results by fire assay. This is evidently a subject for investigation, as either there is something present in the ore which acts upon the gold, or else the ore has been doctored.

C. P. SAWYER ('81) writes from High Point, North Carolina, where he has for some little time been acting as assayer for the Hoover Hill Mining Co., that he has been advanced to the position of assistant superintendent.

WM. W. SHARE ('81) has been appointed Assistant in Physics in place of Dr. M. C. Ihlseng, who has accepted a professorship in the Colorado State School of Mines.

THE common practice in many colleges of conferring the degrees of D.S. and Ph.D. without examination, as an honorary degree, still agitates the minds of those who have earned their doctorate. Resolutions have been passed by the American Philological Association, and recently by the American Association for the Advancement of Science, deprecating the custom, and a committee has been appointed to address a memorial on the subject to the trustees of all colleges. As the matter now stands, fully half the degrees of Ph.D. conferred are honorary.

THE time draws near when the annual circular of the Alumni Association of the School should make its appearance, preparatory to the holding of our Alumni dinner.

It is desirable that our address list should be as perfect as possible, in order to render the labors of the resident Secretary less arduous.

We ask, therefore, that graduates receiving THE QUARTERLY, will inform us of any changes or corrections to be made. We note the following, which have already been communicated:

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1867—John Adams Church, E.M., Ph.D., Superintendent Tombstone Mining Co., Tombstone, Arizona.

1875—Magnus C. Ihlseng, E.M., C.E., Professor of Engineering, State School of Mines, Golden, Col.

1875—Arthur Macy, Superintendent Silver Mountain Mining Co., Ophir, Col.

1875—John Krom Rees, A.B., A.M., E.M., Instructor in Geodesy and Astronomy, and Director of the Observatory, Columbia College, New York.

1876—Francis Newberry Holbrook, C.E., Corralitos, Chihuahua, Mexico.

1877—Henry Gilbert Clark, E.M., C.E., died of typhoid fever at Aurora, Nev., August 2, 1881.

1877—William Kelly, A.B., E.M., Superintendent Kemble Coal and Iron Co., Riddlesburg, Bedford County, Pa.

1878—W. J. Adams, Assistant Metallurgist Batopilas Silver Mining Co., Batopilas, Chihuahua, Mexico.

1878—W. B. Devereux, A.M., E.M., Superintendent Takoma Copper Mining Co., Globe, Arizona.

1878—William Strieby, E.M., Professor of Metallurgy and Assaying, Colorado Springs, Col.

1879—F. B. Stewart, Ph.B., '79, died of typhoid fever at Harlem, N. Y., August, 1879.

1880—Alfred Lockwood Beebe, Ph.B., Assistant in Assaying, School of Mines, Columbia College.

1880—Frank B. Benjamin, C.E., returned from Europe; address, 31 West Nineteenth Street, New York.

1880—William Frederic Brugmann, Ph.B.; address, Chapel Hill, N. C.

1880—Robert Otto Francke, Cuba.

1880—James H. Merritt, Ph.B., Chemist Bradley White Lead Co., 181 Water Street; address, 184 Lefferts Place, Brooklyn.

1880—Juan A. Navarro, Student in Europe.

1880—Frederic A. Potts, Jr., returned from Europe.

1880—Theodore Tonnelé, Ph.B.; address, 48 East Sixty-eighth Street, New York.

1881—Charles A. Andresen, E.M., with Pickard & Andresen, 8 Gold Street, New York.

1881—F. T. Aschman, Ph.B., with A. W. Hale, 71 Broadway, New York.

1881—C. P. Bleecker, E. M., Assayer, Ogden, Utah.

1881—V. M. Braschi, Ph.B., Resident Graduate; candidate for degrees of E.M. and Ph.D.; address, P.O. Box 4,303, New York.

1881—E. R. Bush, E.M.; address, 45 East Nineteenth Street, New York.

1881—P. E. Chazal, E.M., care of Langston, Crane & Co., Atlanta, Ga.

1881—A. L. Colby, Ph.B., Inspector State Board of Health; also Resident Graduate; candidate for degree of Ph.D.; address, School of Mines, New York

1881—E. M. Douglass, C.E., Erie Railroad; address, 143 South Division Street, Buffalo, N. Y.

1881—Edward K. Dunham, Ph.B., Student College Physicians and Surgeons; also candidate for degree of Ph.D., School of Mines; address, "The Florence," Eighteenth Street and Fourth Avenue, New York.

1881—Arthur H. Elliott, Ph.B., Assistant in Chemistry, College Physicians and Surgeons; also candidate for Ph.D., School of Mines; address, School of Mines, New York.

1881—H. Van F. Furman, E.M., Walker House, Salt Lake City, Utah.

1881—W. T. Griswold, C.E., Sinalo and Durango Railroad, Culiacan, Mexico.

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1881—F. A. Hemmer, Jr., Ph.B., Student of Metallurgy, Clausthal, Germany.
1881—C. B. Judd, E.M., United States Deputy Surveyor, Tombstone, N. M.
1881—D. J. Leary, C.E., fourth year Student in Mining Engineering, School of Mines, New York.
1881—Augustus D. Ledoux, Ph.B.; address, Cornwall-on-Hudson, Orange County, N. Y.
1881—W. P. Little, E.M., Resident Graduate, Course in Architecture; address, 60 West Fortieth Street.
1881—W. M. Meserole, C.E., Topographer, Central Railroad of New Jersey; address, 590 Lorimer Street, Brooklyn, N. Y.
1881—Percy Neyman, Ph.B., Student in Chemistry, Berlin, Germany; address, care E. H. Neyman, 97 Maiden Lane, New York.
1881—M. J. O'Connor, E.M., San José Mine, Egan Cañon, Nev.
1881—T. D. O'Connor, Ph.B., Resident Graduate; candidate for degree of Ph.D.; address, 42 West Twenty-eighth Street, New York.
1881—Lucius Pitkin, Ph.B., Chemist, Laurel Hill Chemical Works, Penny Bridge, L. I.
1881—George S. Raymer, E.M.; address, 63 Seventh Avenue, Brooklyn, N. Y.
1881—W. T. Richmond, Ph.B., Chemist, United States Geological Survey, Department Lake Superior; address, 30 West Sixty-first Street, New York.
1881—A. C. Roberts, E.M., Sinalo and Durango Railroad, Culiacan, Mexico.
1881—C. P. Sawyer, Ph.B., Assistant Superintendent Hoover Hill Gold Mining Co., High Point, Guilford County, N. C.
1881—W. W. Share, Ph.B., Assistant in Physics, Columbia College; address, 336 Navy Street, Brooklyn, N. Y.
1881—Chandler D. Starr, C.E., New York, Buffalo, and Western Railroad; address, 103 East Sixteenth Street, New York.
1881—T. B. Stearns, E.M., Metallurgist, Alta Montana Mining Co., Wickes, Jefferson County, Montana.
1881—A. E. Swain, E.M., Sinalo and Durango Railroad, Culiacan, Mexico; address, via San Francisco, Cal., and Mazatlan.
1881—E. G. Tuttle, E.M., Assistant Engineer, Ohio Railroad, Toledo, O.
1881—A. H. Van Sinderen, Ph.B., Student, Columbia Law School; address, 178 Columbia Heights, Brooklyn, N. Y.
1881—H. T. Vulté, Ph.B.; address, 7 Sutton Place, New York.
1881—F. G. Weichmann, Ph.B., Private Assistant to Professor Chandler; also Resident Graduate, candidate for degree of Ph.D.
1881—W. W. Williams, C.E., fourth-year Student in Mining Engineering, School of Mines, New York; address, Wethersfield, Conn.
1881—H. M. Wilson, C.E., Sinalo and Durango Railroad, Culiacan, Mexico.

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THE

SCHOOL OF MINES QUARTERLY.

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THE PROGRESS OF THE METALLURGY OF GOLD AND SILVER IN THE UNITED STATES.

BY T. EGLESTON, PH.D.

THE progress of metallurgy in the United States during the last thirty years has not only been very rapid, but has produced very extraordinary results. Previous to the discovery of gold in California in 1848, the United States produced but a small portion of the world's product of metals, and its people were not known as skilled in the mining or metallurgical sciences. The great coal-fields were known and worked, and iron was produced in what would now be considered very small quantities. The discovery of copper on the upper peninsula of Lake Superior, and of gold in California, showed the want of knowledge of these subjects amongst our people, and undoubtedly gave the stimulus to the energy and enterprise which now makes the United States one of the greatest of mining and metallurgical nations. Twenty-five years ago there was no place in the United States where an elementary knowledge of either mining or metallurgy could be had in any of the educational institutions of the country. To-day a more thorough knowledge of these sciences can be had here than anywhere else. The prospector of these early days has been obliged to give way to the thoroughly trained mining and metallurgical engineer. No finer or better

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examples of what skill and science can do are to be found than the works which have been erected in different sections of the United States within the last fifteen years. In the early days, when prospectors and mineral wands were more or less relied on, "science," which meant education in general, was looked on with suspicion; the demand was for "practical men." But as soon as it became apparent that there was enormous wealth in the ground which could not be extracted by half-educated men with empirical means, science was called in. In the first few years the "scientific cuss," as he was called, was expected to be a mining, metallurgical, mechanical, and chemical engineer, and in all cases where he did not know, to improvise his knowledge; but it was not long before experts in all these branches were to be had, and, in consequence, new methods and new processes have resulted, so that mining and metallurgy have made more progress on this side of the Atlantic in the last thirty years, than had formerly been made in the Old World in more than one hundred. A quality of mind seems to have been produced by the mixture of races on this continent, which, while demanding for the people as a whole the most general information, enables us to easily grasp details, while the eminently mechanical genius of our people leads us to adapt, improve, and invent machinery for every purpose, and thus, by replacing hand labor, to both improve and cheapen the cost of processes.

The gold excitement in 1848 was produced by the discovery of the very rich placer mines on Sutter Creek, Amador Co., California, from which the gold in quite large pieces was easily separated by washing with water in a common pan. Shortly after, the discovery of a number of nuggets of large size,[1] and

[1] A list of the large nuggets is not without interest. One of 3000 lbs. was reported to have been found in the West Indies and sent to Spain in a ship which never reached its destination.

NUGGETS FROM AUSTRALIA.

Sarah Sands, 1850 (?)	223 lbs. 4 oz.	Old Daisy Hill, 1855	73 lbs. 4 oz.
Ballarat, 1853	184 " 8 "	Ballarat, 1855	48 "
" "	135 "	Bendigo, 1852	47 "
Blanche Barkley, 1842 (?)	146 " 6 "	Ballarat, 1855	40 "
Two, 1849	112 "	Bendigo, 1852	28 " 4 "
One, 1851	106 "	Dascombe, 1852	28 "
Maryboro, 1855	88 " 4 "	Mt. Blackwood, 1855	24 "
Fryer's Creek, 1855	84 "		

URAL MOUNTAINS.

Near Miask, Siberia, 1842	96 lbs.	One	27 lbs.
In 1878	78 "	Several of	16 "

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the rumor that others had been found and cut into small pieces to prevent their value being known, caused the wildest enthusiasm all over the United States. Some large fortunes were made among the crowd of adventurers who rushed to California, but no one can know of the misery and disappointment caused by this wholesale emigration. The success of the few was widely published, but the record of the failures was never made. What they must have been is shown by the fact that after the State of California had provided insane asylums for the percentage usual in other States, the authorities were obliged to double their capacity.[1]

So entirely has mining formed a part of the history of the country, that the very words which express the miner's joy or sorrow, his success or failure, have passed into the current conversation of society. To "strike it rich," to "find pay," to "pan out," or to "get down to hard-pan" or "to the bed-rock," indicate to the people of the Western coast their success or failure, their prosperity or adversity, more emphatically than any other words in the English language could do.

It is very doubtful whether the Western States and Territories would have been settled even now but for the fabulous yield of the early discovered surface deposits. California had been more or less occupied for two hundred years by Spaniards and Mexicans. It was known in a general way that gold was to be found either in that country or districts adjacent to it. Shallow placers had been worked with very small results for a long time. Placer gold was discovered and worked in 1841.[2] In 1843 gold-bearing quartz had been worked. But the discovery

PARAGUAY.

One of 50 lbs.

PERU.

In 1730, one of 45 lbs.

UNITED STATES.

California, Calaveras Co., 1854	160 lbs.	N. Carolina, Cabarrus Co., 1838.	37 lbs.
" Sierra Co.	106 "	California, Sierra Co.	32 "
" " 1869	95½ "	" Georgetown, 1865	17 "
N. Carolina, Reed Mine	80 "	N. Carolina, Anson 1829	10 "
California, Sierra Co., 1871 (not authenticated)	64 "	California, Nevada Co.	8 "

In England several nuggets of 40 ounces have been found, but none larger. None have been found in Brazil.

[1] I have this statement from a former Treasurer of the State of California.

[2] *Engineering and Mining Journal*, vol. 32, p. 170.

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of gold in the sluice of a mill depopulated, for the time being, the city of San Francisco, brought a small population from China, and thousands from Mexico; but the country was in a very few months taken possession of by men from the Eastern States, attracted there both by the fabulous stories about the gold and by their love of adventure. Everybody who could dig (and who could not?) became at once a miner. It was not an unfrequent thing, so rich were the deposits, for men who could not earn more than seventy-five cents a day where they came from, to make several hundred dollars between sunrise and sunset, with the rudest appliances, and with a loss of no one knows how much gold beside. This great gain was, however, of but little use to him, because, having acquired it without much labor, no special value was attached to it, and he spent it as freely as he found it. The price of everything in California was, consequently, very high.[1] Trade was carried on as barter, in which "gold dust," as it was called, which was of all sizes, from a nugget the size of the fist to an impalpable powder, answered the purpose of a medium of exchange. In every-day life the fine material was used in the place of coin. It was carried by the miners in buckskin bags which were opened for the merchant to take his own pay by repeatedly taking out what he could hold between his thumb and forefinger. A single pinch from the bag was considered to be equal to twenty-five cents, which was the lowest value recognized; two pinches were equivalent to fifty cents, four to one dollar, and so on. The honest (?) merchant of those days kept his forefinger pressed on a conical button in his waistcoat, in order to make a depression in it, so that the intrinsic value of the single pinch was often equal to a dollar, or even more. This was of no consequence whatever to the miner, who did not condescend to weigh his gold, except in the case of nuggets or of large payments or bank deposits.

At first, only the shallow and very rich placer mines were worked. They required no capital, and but an extremely limited amount of intelligence and skill to work them. So long as they were easily found and the pay abundant, the separation of the gold was only approximative, and no special improvement was made in the processes of extraction. If the sand was very

[1] Flour, $7.50 per pound; butter, the same; a barrel of pork, $210; a bottle of ale, $5; a candle, $3; a tin pan, $9.

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rich and water was not easy to be had, it was simply winnowed in a blanket, and what the wind did not blow off was amalgamated when mercury could be procured, or it was panned in a rough way when water could be had, or was approximately picked out when there was neither water nor mercury. Any method was used without regard to loss, so long as the pay was made rich enough to transport, and could be sold by assay.

By such rude methods only a very small percentage of the gold was extracted, but it would cost trouble to get it all out, so it was left where it was. The miner was then careless of his mode of working, giving no thought to the future, and was extravagant in his habits and methods. In less than five years after its first discovery, the quantity of gold had reached a maximum, and after 1853 began to decrease steadily. As the amount of gold diminished the methods of extracting it improved; the miners became aware that they were losing gold, and little by little they ascertained that this loss occurred in several different ways: either it was not separated from the lumps of sand or clay and was carried off with them, or it was so thin in the pay rock or became so thin in the process of extraction that it was buoyed up and carried off by the water, or was lost because the gold for some reason did not amalgamate. At first he simply called this light gold "float" and the other "rusty," and made no very great account of it. But as it became evident that a very large proportion of the gold in the "pay" was being lost in this float and rusty gold, he became anxious first to ascertain how much there was that he did not catch and what part of this was float—a material so light that it was carried off upon the water, and was thus swept away—and what part was rusty, that is, did not amalgamate with the mercury. To determine this he sought for machines to work with. The miner's pan had up to this time been the tool with which he "got the color," which is the expression used when the gold begins to appear after washing the pay dirt in a pan; but when he found that he must have other appliances, the miner of 1848 took the processes which he found in use by the Mexicans and the Mexican Indians in California, and adopted them bodily. These were, the miner's pan, which he adopted when he first came into the country, and continued to use so long as the placers were very rich; the arastra, and the Chilian mill upon which so much ridicule has been cast

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in later times, and to the principle of which we shall probably have to come back as the gold grows scarcer. The rude methods of concentration associated with these machines were the only methods that he found there, and these he accepted without at first thinking of any improvement.

With the altered condition of affairs every experiment that want of experience or an active imagination could suggest was tried. Every man improvised himself a miner, and felt it his bounden duty to commence where Tubal-cain did, but without his intelligence. The sum of all that is left of these experiments is the sluices, riffles, and undercurrents which are still in use. They alone represent the activity and energy which in the first few years produced hundreds of failures and brought loss and disappointment, and in many cases ruin, to thousands of men. But the necessities of the case had so stimulated the active minds who brought their energies to bear on the subject that the failures were individual, as the loss of wealth of a country which no one owned was not estimated, and consequently was not felt. The country reaped the benefit of all these efforts and all the gains, and in consequence has made greater strides in the way of progress, and more improvement in thirty years than has ever been made in a century by any other nation.

It is impossible to estimate the losses which must have occurred in these early days by such rough methods of working. The "Forty-niners," as they are called, cared very little whether they got out all the gold. What they did care for was to make the greatest present profit not only between the first and last of the month, but between sunrise and sunset of every day. Sunday did not count for a holiday then. It was of no consequence to the miner what he lost, but it was of great importance to him what he gained. He recognized only to-day; he knew and cared nothing about the time when through his slovenly methods gold would become scarce, and if he knew that he was losing gold his regret was for himself because he did not get it, and not that the country was a loser. He knew nothing about political economy, and would have cared nothing for it if he had. He was extravagant in his habits, generous to a fault, careless of his methods, entirely satisfied to let the future take care of itself. By the time that the placers began to grow less productive the population of the country had increased to such an extent that, labor becoming abundant and wages so comparatively low, the necessity of living, which Talleyrand declared not to be

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necessary, became a stimulus towards improvement in the processes and to the effort to save what had been previously considered as not worth saving. More careful washing made men look for a more abundant water supply. The freer use of mercury enhanced its value and set the prospectors searching for mercury mines.[1]

Our present methods, which are being constantly improved, appear extravagant to those who have been trained in the European schools, because they do not take into consideration the fact that with a rate of wages much higher than in Europe, and with transportation at its maximum, their economies would cost more to carry out than they would yield. Hence the foreigner coming to this country has generally to commence his education here by unlearning some parts of the knowledge which he has been taught to consider as his first principles, before he begins to acquire the new ideas which are essential to his successful practice among our mines and metallurgical works.

There is in this country no impediment to progress and improvement, as there is in Europe in what is known as ancient legislation, where antiquated processes have become customs, and their use precedents not to be easily set aside. Their statute books are full of laws preserving the rights of the government, of John Doe and Richard Roe, in certain specific things, and requiring in certain defined cases the following of ancient precedents or the use of obsolete processes and methods, the laws being made with reference to a condition of things which existed perhaps one hundred, three hundred, or even five hundred years ago, and remaining still unrepealed. It is undoubtedly in principle a wise precaution not to allow the investment of capital in mining enterprises without having the whole question both in a commercial and industrial point of view examined by government officers, and the government sanction given or withheld according as in the judgment of its officers it should or should not be given; but it places a great deal of power in the hands of individuals who desire to hold the leading-strings, and does away with the individuality of enterprise which is one of the chief characteristics of our nation. The anxiety to prevent the individual loss in Europe often retards the prosperity of a whole

[1] The discovery of very large deposits of the ores of mercury in California, and the original process invented there for working it, are even more remarkable than the progress made in the working of the other metals. See *Engineering* London, vol. 28. p. 239.

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country, while the precedents as well as the written and unwritten laws of some of the older countries, and the system of centralization in their governments which entrusts everything belonging to the government to corps or to the heads of bureaus at the seat of government, are great impediments to progress. In this country we are at liberty to deal with the necessities of the case as they occur. The impediments to progress are not too much but too little law, not ancient but conflicting legislation. These impediments are, however, incident to any new order of things, and it will not be many years before the last of these difficulties will be removed.

When shallow placer-mining became less remunerative, the question of transportation commenced to be a factor in the question of production. While a miner was earning $100 a day, the fact that he was forced to "pack" every article he purchased for use as well as his gold on a mule to be carried for days or weeks, or concealed his treasure about his person and travelled on foot a considerable distance to invest or deposit his dust, was a matter of very little consequence in view of the large amount so easily earned.[1]

But when he was reduced to "hard-pan" and only earned a little more than was necessary for the sustenance of life, the question both of improved processes and of transportation became a very serious one, and had to be studied very carefully. At this point he either improvised himself an engineer of mines, and with ready Yankee wit devised some process which would save "the color," or transported himself and his "outfit," which in those days meant what he could carry on his back, with his pick, spade, pan, horn or spoon, or more probably all of them, to some locality which was not so far removed from civilization. Others on the contrary sought in localities farther from civilization richer pay diggings.

It was not long before this continued migration produced its results, for while the miner at first searched only for gold and worked for gold alone, when it became scarce and the necessity of living became to him imperative, translating itself into the gnawings of hunger, other things became valuable besides gold,

[1] The safety and almost impunity with which large amounts were carried about the person in the early days of California mining was owing to the rapidity with which the vigilance committees of those days executed justice, which was not only speedy but severe. There were then no lawyers to impede the progress of what was almost invariably real justice, and death was usually the penalty inflicted.

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and he then stopped in his transit from one district to another in search of the precious metal to locate a silver deposit, sometimes in the shape of lead ores, but he never noticed iron or copper. It was not until long after that he made the discovery that copper ores frequently carry gold.

PLACER MINING.

The deposits of gold first worked were called placers. These were at first recognized as the dry and the wet, the dry being either ancient river deposits or recent river beds from which the water was gone. The wet placers were streams which were sometimes large rivers. In these placers the gold was within twenty feet or so of the surface. The dry placers were worked with pans and sluices; the rivers containing the wet placers were dammed and the whole of the water turned into flumes or sluices. The river mud was then sluiced. As it was impossible to keep the river bed dry, both because the dam leaked and because the surface drainage could not be kept out, large wheels called "flutter wheels" were placed in the artificial channels, which pumped the water out of the river bed and furnished a sufficient supply of it to the sluices. Such workings were hazardous and expensive, for if the stream was at all large a single freshet might wash out the dam and the flume which carried the river. To make sufficient room to work and also to get enough ground to justify the expense, the flume carrying the water of the river was never less than two hundred and fifty yards and sometimes nearly a mile long. Such works were very expensive and somewhat hazardous, but sometimes yielded a very large profit. They were constructed by companies during the early days when the miners used less engineering skill and were willing to take more risks. Very few of these river claims exist now, and this kind of mining is already a thing of the past.

As the placers grew poorer the gold was sought deeper, and it was then discovered that there were placers where the depth of the deposit was over three hundred feet, where the gold on the surface sometimes yielded only a few cents to the cubic yard, but was rich on the bed-rock, hence the name of deep placers. Only shallow or surface placers and deep placers are now recognized. It is estimated that two thirds of all the gold produced in the world is taken from placer deposits.

It is quite as impossible to form any adequate idea of the

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amount of gold produced during these first periods of mining as it is to estimate the losses. No attempt was made to make any record at the time. The records of the custom-houses are the only ones made, and are necessarily imperfect; they show that between 1848 and 1854 over $350,000,000 had been recorded, but previous to that time very large sums had been carried away by those going either to Europe or to the East, either in their effects or on their persons, and while the amount carried by an individual was small, the aggregate must have been very large. In the next decade more perfect records were made, which show a gradual decrease, the highest point having been reached in 1853 when the amount recorded is $65,000,000, and the lowest $26,000,000 in 1864, the total amount for the decade being $450,000,000. The next decade to 1874 shows a product of only $220,000,000, the lowest amount reached being $17,000,000 in 1873. This rapid decrease gives a very fair idea of the amount which must have been wasted within the first seven years of the most prosperous time, when the records, tho very imperfect, give an average of $50,000,000 a year, while the year 1873, when the records were most perfect, gave only $17,000,000. From 1873 to the end of 1880 the total product of the State of California was $135,113,000; the lowest annual product being $15,000,000 in 1877, and the highest $18,000,000 in 1874. The following tables, showing the amount in each year, have been prepared for me by the Director of the Mint at Washington.

Year.	Gold in Dollars.			Year.	Gold in Dollars.		
	California.	Other States.	Total Gold.		California.	Other States.	Total Gold.
	$	$	$		$	$	$
1848	10,000,000	.....	10,000,000	1865	28,500,000	24,725,000	53,225,000
1849	40,000,000	....	40,000,000	1866	25,500,000	28,000,000	53,500,000
1850	50,000,000	.....	50,000,000	1867	25,000,000	26,725,000	51,725,000
1851	55,000,000	.....	55,000,000	1868	22,000,000	26,000,000	48,000,000
1852	60,000,000	.....	60,000,000	1869	22,500,000	27,000,000	49,500,000
1853	65,000,000	.....	65,000,000	1870	25,000,000	25,000,000	50,000,000
1854	60,000,000	.....	60,000,000	1871	20,000,000	23,500,000	43,500,000
1855	55,000,000	.....	55,000,000	1872	19,000,000	17,000,000	36,000,000
1856	55,000,000	.....	55,000,000	1873	17,000,000	19,000,000	36,000,000
1857	55,000,000	..... .	55,000,000	1874	18,000,000	15,490,902	33,490,902
1858	50,000,000	.....	50,000,000	1875	17,000,000	16,467,856	33,467,856
1859	50,000,000	.....	50,000,000	1876	17,753,000	22,176,166	39,929,166
1860	45,000,000	1,000,000	46,000,000	1877	15,000,000	31,897,390	46,897,390
1861	40,000,000	3,000,000	43,000,000	1878	15,260,000	35,946,360	51,206,360
1862	34,700,000	4,500,000	39,200,000	1879	17,600,000	21,299,858	38,899,858
1863	30,000,000	10,000,000	40,000,000	1880	17,500,000	18,500,000	36,000,000
1864	26,600,000	19,500,000	46,100,000				

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It is remarkable that while up to 1860 there was almost no gold produced in the Eastern States, and that while in that year only $1,000,000 is reported, between 1860 and 1870 the amount produced gradually increased, until in 1866 it exceeded it, and only fell so as to equal it in 1870. In the next decade, from 1870 to 1880, the largest amount produced in all the other States except California was $35,946,360 in 1878, and the lowest $15,490,902 in 1874; with the exception of 1874 and 1875 the Eastern States produced more than California, and in 1878 more than double the amount produced there. The total yield of gold in the United States in the year 1880 was only $36,000,000, or less than that produced by California alone up to 1862, with the single exception of the year 1848.

It is extremely difficult to ascertain the amount of gold contained in the world, and it may be said that the best estimates from the most reliable authorities are but the merest approximations. It is supposed, however, that the entire amount of gold in existence at the present time is not over $7,000,000,000. As the value of a cubic yard of gold is $9,000,000, if all this gold were melted into one mass it would contain about 700 cubic yards, which would make a block about 25 by 25 by 31 feet, or less than the cubical contents of an ordinary small city house.

Long before the fact that the gold was decreasing became known to the miners, it was evident that some new system would have to be used to extract it. The miner's pan, a household utensil, was used not because it was the best adapted to the work, but because it was the most available. It was an exceedingly rude apparatus. It was made of sheet-iron from four to five inches deep. It was filled two thirds full of dirt, and put into a hole about one foot deep filled with water, and the contents of the pan well stirred with the hands; the pan was then taken in both hands, inclined slightly outward, and shaken so as to give a circular motion to the contents. The fine material flows in this way over the sides. When this has been done long enough, the stones are removed, any lumps of clay are broken with the hand; then depressing the outer edge, it is shaken until nearly everything but the black sand is out of the pan. This is separated by blowing. It required great skill, and gave a large loss in gold. It is now rarely used except for an assay for color, for which the shovel can also be used. The pan was used for a long time after other machines were introduced, but it soon

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became necessary to have other implements. As an instrument for concentration, it was succeeded by the cradle or rocker, which was not unlike a child's cradle, whose maximum capacity was 5 to 6 cubic yards of earth a day. It required two men to work this cradle efficiently, one to rock and pour on water, and one to bring the pay dirt and the water, whose weight was at least three times that of the dirt. The pay dirt was thrown upon a screen whose object was to separate the large stones and to help to break up the clay; the purpose of the water was to take up the finely divided particles. The concentrates are then panned. A pan contains about half a cubic foot of pay dirt. A one-man rocker could concentrate from 100 to 150 pans a day, and a two-man rocker twice as much. It is a very slow machine, must always be placed near the water, no matter what the distance of the pay dirt is, and even when used with quicksilver loses a very large amount of fine gold and all the float gold.

The cradle was succeeded by the "long Tom," which for nearly two years seemed a great improvement. It consists of a rough trough or launder eighteen feet wide at the upper end and thirty feet at the lower, and twelve feet long. The lower end is terminated by a screen of iron whose edge is so high that the water does not flow over it, but drops with its contents into a trough below provided with riffle bars and mercury. The pay is thrown in at the upper end and mixed with water; what passes the screens comes in contact with the mercury and is caught, but much of the fine and all the float gold is lost, tho it is a better machine than the cradle. As many as four men can work at a Tom, but it is now rarely seen.

At the same time a puddler was used, which was a barrel cut in half or a rough wooden box six to eight feet square, and twelve inches high, with an inch and a half hole four inches from the bottom. The pay was thrown into this and broken up and mixed with the water until it was all in suspension. When the plug was removed, the thin mud was allowed to run out and the operation commenced again, and so on. The gold was collected in the bottom. The puddler never was used in this country except in small claims where water was scarce, but has been very successfully and extensively used in other countries where water was not abundant.

The next improvement was the sluice, which, altho it had been formerly used elsewhere, is as much a California invention

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as if it had never been used before it was invented here. It consists of a trough or launder made of rough one and a half inch boards sawn for the purpose, and nailed together without any pretence of making them fit, as it becomes tight by the swelling of the wood and the filling up of the cracks. The sluice is composed of a series of boxes, as they are called; each box is twelve feet long, sixteen to eighteen inches wide at one end, and twenty to twenty-four at the other. The height of the sides varies from eight to twenty-four inches, according to the kind of material to be used in it. The narrow end of one box fits into the wide end of the other. The length of the sluice is estimated in boxes; it should not be less than fifty feet long, and often consists of hundreds of boxes put together supported on rough trestles. The inclination is regulated according to the necessities of the case, and is called the "grade," which always has reference to the box. It is usually eight inches for a minimum and thirty inches for a maximum in twelve feet, or the length of a box. It is generally uniform, but sometimes is made to conform somewhat to the lay of the ground. The tougher the dirt, the longer and steeper the sluice must be. With a rapid descent the dirt is also much more easily separated, but the greater is the loss in gold. The boards are not only rough on the bottom and sides, but to prevent too rapid wear, and also to help to catch the gold, strips of wood are fastened in the bottom of the box, either in the direction of its length, which is most usual, or across it. These pieces of wood are from two to four inches thick, and from four to six inches wide. When longitudinal they are six feet long, so that two sets go in a sluice. The riffles are wedged in so that they can be easily removed for a clean-up or for repairs. The bottom of each box is thus filled with a screen of rectangular cavities which are the width of the distance between the riffles, and have their length and depth. Here the quicksilver which is put in at the head of the sluice rests and catches the gold as it sinks in the more or less rapid current. The water is generally made to run two inches deep over the riffles. The pay dirt is thrown in with shovels. The first dirt, which is always poor, goes to fill up the spaces between the riffles. The water, washing over this, washes out the earth and clay, the sand and gravel, and even the stones. No mercury is added until after the sluice has been running about two hours. It is then put in at the head and finds its way down the sluice, most of it being caught not far from where it is put in. One man can throw in

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from two to five cubic yards of dirt a day. The number of men that can work depends on the length of the sluice and the lay of the ground. Sometimes when the earth to be washed contains large stones, an undercurrent is used. The end of the sluice is then open. It terminates with a grating of iron bars long enough to allow all the water to pass with the fine material through the grating, but allows the stones to roll out on the ground. The whole of the water and dirt is caught in a short sluice below and discharged into one parallel with the main sluice, which is thus continued. This undercurrent, tho sometimes used with the ordinary sluice, is an indispensable part of the plant for hydraulic mining. The sluice is simple in construction and use, requires but little outlay of capital, and is very effective. Unlike the outlays usually made in mining, the whole of the plant in use can be readily taken down and transported to another claim when the first is washed out, and set up there as effectively as if it were new. There is one precaution that must be taken with it, which is that lumps of clay must not be allowed to travel along it without being broken up, for they are liable to pick up the gold as they roll and carry it off with them.

Getting the amalgam out of the sluice is called cleaning-up, and the time between one clean-up and another is called a run. The length of a run will depend on the richness of the deposit, but is usually from six days to two weeks, occasionally longer. A clean-up occupies about half a day, and is usually done on Sunday. To do this the water is allowed to run after the dirt is no longer thrown in, until it is quite clear; six or eight sets of riffle bars are then taken out at the head of the sluice, and the material washed down, while the amalgam is caught at the head of the next riffles. This is taken out. The next set of riffles are then taken out, and so on. The excess of mercury is strained from the amalgam by twisting it in a buckskin bag, and the rest is driven off by heat.

HYDRAULIC MINING.

For many years the sluice was used to work nearly the whole of the placer gold of the country, and it is still the most available way for persons of small capital to treat the shallow placer deposits. It must always be looked upon as a process of great historical interest, for out of it hydraulic mining grew, which

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is one of the most marvellous achievements of modern engineering skill, to which the State of California is more indebted than to all other inventions of mining and metallurgy put together There are localities where poor, shallow placers are found, where water is scarce during most of the year, but abundant at certain seasons, and where the grades are heavy. For these placers another sluice was used, known as the ditch or ground sluice. A small ditch is cut through the placer and the water turned into it, the first object being to deepen and enlarge the ditch to the proper size. When this has been accomplished, the banks are pried into the stream. No mercury is used, but cobble-stones are thrown into the bottom of the ditch so that the gold may settle between them. The effort is to concentrate the gold in the dirt and then work it up in a short board sluice.

These two methods put together were the germs of hydraulic mining now so extensively used in all parts of the world, not only for gold mining, but for the removal of dirt from other ores. It appears singular that the name of the man who really invented the most remarkable process of this century should be as lost to history as if he had never existed; but in the struggle for existence, as the shallow placers grew poorer and poorer, and the gold was found at constantly increasing depths, the man was lost sight of, while his work has now been so perfected that it is one of the marvels of the union of modern engineering skill with capital. The process of hydraulic mining was invented in the spring of the year 1852, on the Yankee Jim claim in Placer Co., California,[1] where an enterprising miner, finding that he was not making sufficient money, began working his claim with a shallow ditch in the side of a hill leading to an ordinary barrel, from the bottom of which a cowhide hose was carried and discharged by means of a tin pipe against the bank, and he thus became the father of one of the greatest of modern inventions, hydraulic mining.

There is another kind of placer which deserves notice, both because it is interesting in itself and because it has been the source of disappointment and loss to so many. This is the beach gold, which occurs between Point Mendocino in Northern California and the mouth of the Umpqua River in Southern Oregon. The cliffs along the ocean front seem to be the remains of an ancient river deposit. They contain gold, and where

[1] *Engineering*, London, Eng., vol. 24, p. 353.

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washed by the waves often show the shore for miles glittering with it. It is very uncertain, however, for what appears to-day is washed away to-morrow. No dependence can be placed on finding in the morning the deposit of the day before, so that all haste is made to carry the sands which are rich enough to some safe place inland to be washed and amalgamated. The beach is very narrow, and when the waves are high they wash against the bank. The gold is washed out with the heavy sand, and as the particles are very fine, it is carried down to near the low-water mark. When the ocean is still, sands of variable richness can be collected, but the waves are often so high as to wash all the sand away to the depth sometimes of six feet, and leave the bare rocks exposed. So changing and shifting is the value of the deposit that it has to be examined every day, and the washing of the following day may sometimes be six miles from that of the previous one. Such sands as these must be very rich to make it possible to run the risk of washing them and to bear the transportation to fresh water, as salt water is of too high a specific gravity to work with, for which reason the attempt to work these deposits by dredging has not been successful. It has been proposed to bring water from a distance and turn these mines into hydraulic placers washing down the banks, and depending on them for their profit, leaving the beach sands as an accessory. The tails would not then be a question of importance, but the economic results of such an enterprise would be very doubtful.

After the introduction of sluicing either on a large or small scale, the pan, the cradle, and the rocker were very rapidly abandoned to John Chinaman, who always succeeded in living off the placers that had been abandoned by the "honest miner," who, however, never had any compunction, if he found the Chinamen were tolerably prosperous, in jumping their claims and driving poor John off to find some other place for his enterprise and cheap labor.

From hundreds of dollars a day, by a gradual decrease stretching over a period of from fifteen to twenty years, the gold became scarcer and scarcer, until now it has become necessary with improved appliances to work in the deep placers material which contains from 3 cents to $1.25 per cubic yard.[1]

[1] In the spring of 1880 I was informed by the President of the North Bloomfield Mining Company that they were now working with a profit gravel containing only three cents to the cubic yard, when only a few years ago it was considered a marvel to be able to work that which yielded ten cents.

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No better evidence of the progress that has been made in the working of placer claims can be had than the comparison of the cost of working them by the older and by the more recent processes. Adopting four dollars per day as the wages of a skilled miner, the cost of working a cubic yard of gravel as given by Phillips is—

With the pan	$20 00	With the long Tom	$1 00
" " rocker	5 00	" " hydraulic process	02

In the very early days the capital required by a miner consisted of a pick, shovel, a horn and spoon and a pan, two stout hands, and a valiant heart; but as the placer mines grew poorer this capital was no longer sufficient. It can readily be understood that such capital as was required in the early days did not necessitate any permanent location nor any very high order of intelligence for its use, and hence much of the surface was dug over and simply rendered difficult to work further without extracting more than a tithe of the wealth contained in the ground, merely because the miner had no permanent interest in one spot more than another, and because his capital was entirely a rolling and not an invested one. Every man then was in business for himself, but as the gold grew scarcer, first money, and then intelligence and capital became factors in the equation, so that the capital required to work any one of these claims became larger and larger, until to-day it is estimated that the plant of the North Bloomfield Mining Company has cost not less than $3,000,000, and their works are marvels both for the originality and engineering ability displayed, consisting of dams 90 to 100 feet high, ditches, pipes, and sluices many miles in length, and every hydraulic appliance which engineering skill and capital can add to their works.

These deep placers are the deposits formed in the beds of ancient rivers which have since been so covered by recent accumulations, or cut across by modern valleys of erosion that without a careful survey no one could recognize table-lands twenty miles distant as part of the same ancient river deposit. In some cases the gold-bearing material has been covered by beds of basalt one hundred and fifty feet thick.[1] In many cases the erosion

[1] A map of the basalt-capped deposits of the North Bloomfield Co. is given in the Report of the U. S. Mining Commissioner for 1875, p. 116.

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has been such that the bed of the old river is now one hundred or even two hundred feet above the surrounding country.

To work these deposits, careful surveys of the whole country must be made so as to be able to reach with a tunnel the lowest point of the bed-rock, which must be determined by sinking shafts upon it. The cost of this preliminary work may often be more than $100,000, and instances have been known where from want of proper judgment in the outset the whole of this sum has been lost. The location of the tunnel must be such that the pay dirt can be washed through it, and that it may form an outlet for all the material which is deposited after the extraction of the gold. Its construction involves the building of miles of sluices to catch the gold and carry the dirt away; the damming of streams to save the winter's water supply; the storing up of billions of gallons, and conducting it in ditches, flumes, and wrought-iron pipes, sometimes forty, fifty, or even a hundred miles in length, the ditches alone costing in some cases from half a million to a million of dollars, and involving constructions which are marvels of lightness, strength, and engineering skill. The following table[1] gives a fair idea of the size and cost of the ditches in California:

	Length Miles.	Capacity in Miner's Inches.	Grade. Feet per mile.	Size in Feet.			Cost.
				Top.	Bottom.	Depth	
Smartsville ditches	.....	5,000	9	8	5	4	$1,000,000
Eureka Lake and Yuba ditches	163	5,800	.....	....	....	.....	723,342
N. Bloomf'd ditches and reservoirs	157	3,200	12 to 16	8½	5	3½	708,841
South Yuba ditches	123	7,000	8 to 13	6	....	4 to 5	.....
Milton ditches and reservoirs	80	3,000	12 to 15	6	4	3½	391,575
Spring Valley and Cherokee	52	2,000	.....	5	....	3½	.....
Hendricks	40½	.....	6 to 12	5	....	2	136,150
Blue Tent	32	18,000	10	8	6	4	150,000
La Grange	20	27,000	7 to 8	9	6	4	500,000

This water is discharged through iron nozzles with a velocity of one hundred and fifty feet per second, and at the rate, in some instances, of 4,220,000 cubic feet in twenty-four hours, against a bank from 250 to 300 feet high, and washes the earth into wooden sluices paved with rock or wood. To make the action of the water more effective, the bank is mined and fired, single blasts of from 1500 to 2000 kegs of powder being made.

[1] Burchard, "Production of Gold and Silver in the United States," p. 318. Washington, 1881.

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As everything in the bank must come down, huge cranes with booms ninety feet long worked by hurdy-gurdy water-wheels are set up to lift the boulders,[1] under-currents[2] to catch the gold, grislies[3] to carry off the stones, drops[4] to break the materials up. The sluice itself has to be paved with stone or wood[5] and furnished with branches, so that one part may be repaired without detriment to the other. Every part must work harmoniously with the other parts, and must be adapted to the lay of the ground, and every possible resource in the surroundings made use of for its successful working. Individual enterprise could do little or nothing with such claims, but the constant and large returns show that the immense outlay is fully justified.

The gold is caught in mercury put into the sluices between the pavement and riffles. The greatest difficulty is not so much to catch the gold as to get rid of the tailings or material that has been treated. This involves the construction of miles of tail-sluices and the destruction of land and of streams by depositing on and in them stones and sand to great depths, but it saves for the use of the country the very large amounts of gold deposited in exceedingly small quantities in the ancient river-beds of California. No one who has not visited these mines can have any idea of the devastation produced by this washing away of hundreds of acres of surface and hundreds of feet in depth of the ground of these gold-bearing districts. It will be many years before this question of local devastation will need to be considered, but the filling up of the rivers and streams is engaging attention now.

The cost of producing one Troy ounce of metal is given below:[6]

	La Grange Co.	No. Bloomfield Co.
Water	$1.43	$2.09
Labor	6.85	3.93
Materials	1.81	0.88
Explosives	....	0.98
Blocks and lumber	....	0.50
General expenses	0.94	0.70
Contingent expenses	0.26	....
Taxes	0.09	....
	$11.38	$9.08

[1] *Engineering*, vol. 23, p. 58. [2] *Engineering*, vol. 24, p. 487.
[3] *Engineering*, vol. 25, p. 20. [4] Ibid. [5] Ibid.
[6] The value of the metal was $18.53 per ounce.

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The height of the bank washed down and the yield for several mines is given below:[1]

	Height of Bank in Feet.	Yield per Cubic Yard in Cents.
Smartsville Claims, Yuba Co.	112	19.5
Blue Tent, Nevada Co.	180	15
North Bloomfield, Nevada Co.	180 to 260	4 to 6.5
Gold Run, Placer Co.	200	4.8
Columbia Hill, Milton Co.	100	4.33
La Grange, Stanislaus Co.	18 to 100	2.5 to 15.5
Patricksville, Stanislaus Co.	40 to 60	4.33 to 18.5
Dardanelles, Placer Co.	150	13

The cost and yield per cubic yard of some of the mines is given below:[2]

	Cost.	Yield.
Roach Hill	$0.06	$0.60
Richardson	0.03	0.15
Iowa Hill	0.025	0.71
Independence	0.02	0.25
Wisconsin	0.02	0.125

The cowhide hose used at first soon became rotten and burst. This was succeeded by one made of heavy duck from four to ten inches in diameter. This was made sometimes of one, sometimes of two thicknesses. Such a hose will bear a pressure of fifty feet, but no more. To make it stronger it was surrounded by iron rings two inches wide and three inches apart, which were held in position by four ropes distributed evenly in the diameter. Such a pipe was called a crinoline hose, and would support a head of from 150 to 200 feet of water. This was subsequently abandoned for the sheet-iron pipe now generally in use. The profits of this kind of mining do not depend so much on the yield of the pay dirt as they do upon the cost of the water, the expense of getting rid of the tails, and the facility of working which depends on the lay of the ground. A claim well situated can work a much poorer gravel with a profit than one less advantageously placed.

TREATMENT OF GOLD QUARTZ.

As the placers grew poorer and the search for other sources of gold became active, the prospectors soon found gold in veins,

[1] Burchard, "Production of Gold and Silver in the United States," p. 318. Washington, 1881.

[2] *Engineering*, vol. 25, p. 59.

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and these were then explored and worked. It was usually found in a hard rock which was ascertained to be quartz, but when it was afterwards discovered in other rocks no attention was paid to the correct name of the stone. The gangue of the gold was always called "the quartz," no matter how hard or how soft the rock was, or what its chemical composition might be; the "quartz," if it was quartz, was said to be hard, if it was slate it was said to be soft. To "get" the vein rock required capital and a much higher degree of skill than had as yet been required for the working of the shallow placers. Deep quartz mining could not be carried on by individuals, and mining com panies were formed to mine the quartz and separate the gold. At first these were all undoubtedly legitimate enterprises, but it was not long before some men found or thought they found a more expeditious road to wealth in mining shares than in mining quartz—a practice which very soon brought discredit on all kinds of enterprises in the State of California.

To get the gold out of the vein-rock it had to be crushed. This was done in the very early days before mining companies were known, with a large rock bound to a pole supported on a crutch so as to have a long purchase. The rock was raised by one man and allowed to fall on the ore, while another kept the pieces of ore from flying away, with a stick of wood. It was not long before this rude Mexican hand labor was replaced by the arastra, which was a hearth or bed of uncut stones arranged in a circle from ten to twenty feet in diameter, with a curbing two feet deep on the inside, over which large stones were dragged by a single mule. It was necessary to run this machine for at least a week, and sometimes for two or even three times as long, to make it worth while to clean up. The joints between the stones were so open that the mercury and amalgam settled down between them so that the whole bed had to be dug up, the earth carefully collected and washed, and the hearth replaced before a new charge could be made. The hearth was then improved by making it of cut stones with very close-fitting joints laid in cement. To the upright part, one arm or two arms at right angles to the diameter of the bed were placed, and to each end of the arms stones weighing from four hundred to five hundred pounds were attached by chains so that the forward end was about two inches above the hearth while the other end dragged on it. One mule was counted for each stone, so that there were one- or two-mule arastras. To make the charge for

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an arastra ten feet in diameter, five hundred pounds of quartz broken by hand to the size of a pigeon's egg was put in, and the mules driven for four or five hours. Water is introduced to make a paste of the consistency of cream, and then quicksilver is added and the mule driven for two hours more. The paste is now thinned with water, the mule driven slowly for half an hour, the mud is run off, and another charge introduced. Four charges can be made in twenty-four hours, but two are generally all that are made, so that half a ton per day is about the limit of capacity of a ten-foot arastra. The clean-ups are easily and frequently made, so that this machine is a much better one than the first, which was built for very rude work.

It is quite easy to see from this description where the idea of pan amalgamation originated. It needs but a few mechanical appliances to be added to have the description of the pan as now used. The arastra is still a useful article, and as it costs next to nothing to erect, it may be used as a prospecting tool. The principle on which it is constructed is an excellent one, the grinding perfect. It gives a large percentage of the assay value, but is too slow for use where large quantities are to be treated.

The Chilian mill was used about the same time, and consists of a circular bed like the arastra, but the ore is crushed by two large circular wheels made either of stone or of cast-iron, which roll around on their edges. The methods used are the same. The machine is more expensive than the arastra, and does not do its work any better. It was not long before the eager and impatient miner found that the capacity of the arastra and of the Chilian mill was not sufficient for the profit which he wished to get out of his ore, and he looked to some of the crushing machines used in Europe to increase the capacity of his mill, and at the same time his hoped-for profit. The stamp-mill seemed the only one likely to be of use, and this was adopted. There were then but few mines whose output would justify the erection of stamps, but where it would do so they were built. In other cases the miners were obliged to depend either on the stamps of some neighboring mine being sufficiently at leisure to do other work than their own, or to mills being erected independently of the mines to do whatever work was brought to them. To distinguish such mills from the others they were called "custom mills." They work for everybody and at a fixed price, reserving the tails for themselves, which they usually take care should be rich.

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The lumbering German stamp with its wooden stems eight inches square, its complicated and cumbersome cam shaft, and its inefficient mortar, was first built. Then some Cornishmen suggested the idea of the Cornish stamp, which, with its rectangular iron stems but inefficient mortar, was quickly put up, and was for a short time such an improvement that it was considered to be the *ultima thule* of crushing. This gradually grew into the rotating California stamp[1] and its adjunct the Blake's crusher, with its increased capacity and much better mechanical appliances, which is now almost exclusively used in California. The stamp with its head weighs from 500 to 1000 pounds. Its capacity is, for very hard rock one and a half tons, and for very soft rock four tons, in twenty-four hours. The average will be about two tons. It is built in batteries of five stamps each, so that each battery may be counted for ten tons in twenty-four hours—a capacity which, considering the cost, is very limited. It is in universal use in California, where no other means of crushing has been able to compete with it. In other parts of the country it has grown into the Ball's stamp,[2] a very large and heavy rotating stamp, in which the force of the blow is increased by the pressure of steam to such an extent that a single machine with one stamp-head is capable of crushing one hundred and ten tons of the hardest rock in twenty-four hours, and thus a single machine becomes equal in effectiveness to a sixty-stamp mill, which is the size of some of the largest mills in the West. No investigation has yet been made to ascertain the comparative efficiency of the use of power in these two machines. The Cornish, California, and Ball stamp are working side by side on Lake Superior, and it is to be hoped that experiments to settle the question of efficiency will be made. There are several other crushing machines which have lately been introduced, but none of them have been fully tested, tho some of them seem to be of great promise.

But it was not sufficient to crush the rock, the whole bulk of the crushed material had to pass over or through mercury in order to extract the gold and silver; and the quantity of this rock was so large that it produced losses in mercury so great as seriously to diminish the profits. Amalgamated plates and some free mercury were placed in the mortar, but this, unless

[1] "Stamp-Mills of California," *Engineering*, vol. 30, p. 19.

[2] *Metallurgical Review*, vol. 2, p. 285.

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the pieces of gold were large, could never catch more than eighty per cent of it, and when the pieces were small not more than seventy per cent. If the gold is associated with silver as in some of the Nevada mines, but very little of it is caught there, and the amalgam so produced is so light and spongy that it is liable to be carried off; so that where gold ores contain much silver, lead, or antimony, battery amalgamation is unadvisable. There are many gold-mills where it is never used, tho some of the best-conducted ones do use it. There must therefore be a sluice at the end of the splash-box to catch the fine particles coming from the mortar. The sluice is thus, as an appendage to the battery, as important as the battery itself. It serves a very different purpose from the placer-sluice, where the dirt is to be transported, the clay broken up, and the stones carried along so as to allow the gold to sink where the mercury is. The battery-sluice has neither to deal with stones, dirt, nor clay. It has to treat much less material; it is therefore much shallower and has a lower grade. When the ore contains copper and iron pyrites, these can by proper treatment be concentrated by it and saved. These battery-sluices are used with amalgamated copper plates with transverse riffles in which mercury is placed, and with the different kinds of blankets. Great attention is being given just now to iron riffles, not only for battery but for placer sluices. The experiments made give reason to hope for a great saving of the precious metals by their use wherever sluices are used. In some localities the battery-sluice is used in connection with such machines as the Attwood's amalgamator[1] and the Eureka Rubber,[2] but the tails are still rich. Some one suggested the use of cowhide with the hair on and the grain of the hair turned against the current, over which the tails were allowed to flow in order to catch the heavy materials, while the lighter ones were carried off by the stream; and out of this grew all the different styles of stationary and revolving blanket-sluices. Still the tails showed by assay that they were rich, and it was then found that the gold was contained in iron pyrites. It is very remarkable that at this period when Mexicans with their arastras and slow but extremely simple processes could make $50 to $60 a day, the best stamp-mills with the most efficient machinery, working on the same rock from the same vein could not recover

[1] *Engineering*, vol. 31, p. 247. [2] Ibid., p. 324.

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from the ore more than $15 to $20. In some instances, with the best modern machinery an ore yielding by assay $700 to $800 in gold did not yield more than $20 to $30 when stamped and treated with the usual appliances. The slower but more perfect process of the arastra had brought the pyrites into such a fine state of division, and had by constant abrasion of the stone rubbed it so bright, that the "quick" took it up, while the more rapid process of the stamp did not. As soon as it was thoroughly understood that the pyrites contained the gold, concentrators of different kinds, with or without buddles or keeves,[1] were used to catch it, as no smelting process could be used in the localities where the pyrites was found, and the gold in the pyrites could not be separated with mercury.

The tails are therefore generally kept to be put through a series of concentrating machines whose object is to catch the pyrites and possibly some amalgam, but saves none of the gold which has escaped as float. It is a question of grave importance how this gold may be caught, or, better, how to prevent its getting into the condition in which it cannot be caught. The miners call the gold that escapes the mercury "float" and "rusty" gold. That some gold exists in the ore in such fine particles that it will float seems undoubted; against this there is no remedy. It is also true that the heavy stamp falling on the ore does make float-gold of some of the precious metal not in that state in the ore; but this is not the principal source of loss. If a piece of pure gold which amalgamates readily is pounded with a hammer on a smooth anvil, it is very soon put into the condition in which mercury will not touch it.[2] I have had such a piece of gold in contact with mercury for more than a week without amalgamation. From this condition the gold can readily be recovered in the laboratory, but it is doubtful if it can be saved in the mill. Something must be done to avoid it, as there seems to be no doubt that some part of the gold escapes amalgamation from this cause. But there is gold which is really rusty, not covered with a coating of oxide of gold, but of something which prevents contact with the mercury. Absolute contact is necessary to amalgamation, and the thinnest film between the two will prevent it. One of these coatings is oxide of iron, which does not occur very often and is very easily removed by abrasion, and

[1] *Engineering*, London, vol. 31, p. 404.

[2] *Trans. Inst. Min. Engs.*, Feb. meeting, 1881.

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another is said to be silica; grease from the stamp, or which may be in the water used, will also produce it. I have shown also that a small amount of sulphuretted hydrogen or sulphhydrate of ammonia will produce exactly the same effect, leaving an impalpable greasy film on the outside which prevents the action of the mercury. In other words, a dirty mill, water which is not clean and not carefully protected from drainage, will affect the gold. No one has until now thought it necessary to call attention to this subject.

I have elsewhere shown [1] that there are many other interesting facts relating to the metallurgy of gold which have escaped observation. How far they may affect its extraction from the ores is yet to be seen. It seems, however, certain that we are creating some of the difficulties, and that some other machine than the stamp will have to be used for pulverizing gold ores.

The cost of the treatment of a gold ore, including the mining, varies very much and depends on a great variety of circumstances, such as the hardness of the rock, cost of transportation, price of labor, etc. It may vary from one to ten dollars per ton. It is usually higher in custom mills than in those mills owned by the mine, where the quantity treated is very large. In such mills the cost of crushing, varying with the rock, will be from one to two dollars per ton. In the custom mill where only a small quantity is run it may be as high as five dollars for the same ore. The yield of the ore and the consequent profits are very variable. Ores yielding as low as five dollars have been treated in a large way with a profit, but so much depends both on the nature of the ore and on local circumstances, such as the management of the mill, that it is quite impossible to say how poor an ore could be worked. Perhaps no one thing has been so great a stimulus toward the perfection of the gold processes of California as the discovery of the mercury-mines there.[2] The demand for a large quantity of mercury so stimulated the rival companies that they improved their process to cheapen their production, and found themselves with such active competition that the price of quicksilver has fallen to less than one fourth of what it was, so that the free use of mercury is no longer feared in the poor mines, and the ominous question of the loss in mercury does not loom so high nor figure for so large a part of the cost in the processes as formerly.

[1] *Trans. Inst. Min. Engs.*, Feb. meeting, 1881.
[2] *Engineering*, vol. 28, p. 239.

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For a number of years all the metallurgical losses of gold-mining were attributed to sulphurets, and processes for working these were invented without number, most of which have died out long ago. The beautiful process of Plattner[1] for roasting these sulphurets and then extracting the gold by chlorine was adopted and improved upon, and for a long time seemed to have solved the question, but little by little it was ascertained that there were certain substances contained in the gangue of the rock, such as lime and magnesia, and certain other substances which might be associated with the gold, such as lead and zinc, which would be attacked by the chlorine, and that there were circumstances in which after the gold was in a soluble form in the tanks it might be again precipitated in the insoluble material of the ore, and thus be lost. Recent experiments have been made with nascent chlorine with this gas under pressure, and other modifications which give promise for the future. It is apparent, however, that Plattner's process in any or all of its modifications does not cure all the evils, because it does not cover every variety of case, but is only applicable to certain ores in which there is nothing but the gold in the ore which would be attacked by the chlorine, and nothing which would prevent its acting on the gold.[2]

If the ore contained any silver this would be attacked, and a coating of insoluble chloride of silver would be formed over the gold; this would prevent further attack by the gas, and not only would the silver be entirely lost, but any particles of gold contained in it would also be lost. A very careful dressing might separate some of the gangue attacked by the chlorine, but it could never separate the whole, and any part of it remaining would be a source of loss; so that as a general rule it may be stated that when the ore contains anything but gold which the chlorine will attack, the process is not applicable. Besides, Plattner's process depends upon delicate chemical reactions. If, for instance, any of the sulphate of iron resulting from the roasting is left in the ore, as soon as the gold has been put into a soluble condition and is leached with water, a part of the gold dissolved is thrown down by the sulphate of iron. Certain organic compounds produce the same results. The gold thus previously rendered soluble is then precipitated in the

[1] *Engineering*, London, vol. 24, p. 119.

[2] Clay becomes diluted in the water. The earthy particles in suspension settle on the small flakes of gold and prevent the action of the chlorine.

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filter of gravel on the bottom of the tub and is lost.[1] I saw such an accident at Grass Valley, and on examining the sand of the filter found it very rich in gold. It is not very wonderful, therefore, that a process which demands such nice working and depends on exact chemical reactions does not succeed very well where there are no trained metallurgists. I have known an expensive plant abandoned and the process brought into great disrepute for one of the reasons given above, when if there had been a trained metallurgist in charge there is every probability that the defect would have been remedied and the process would have succeeded.

It sometimes happens, too, that where the process is otherwise applicable, certain substances held in solution in the water of the district, if the gold already dissolved in the water is allowed to remain for a short time in the tanks, and sometimes by simply filtering through them, cause the gold already dissolved to be precipitated on the filter and thus to be lost. In some cases where the works might have been successful had there been only one, there were so many competing for the product of the district that they were obliged to lie idle more than half the time or pay a higher price for the concentrates than they were worth, in order to keep at work.

As a result of the knowledge that gold could be recovered from the tails, there followed a series of concentrating-machines, of blanket-sluices, of different kinds of amalgamators, of pans for grinding and amalgamating, and of single machines for doing a dozen things which require to be done each by a separate machine in order to be well done.

For a number of years the idea that gold and silver could be concentrated by means of smelting had not occurred to the people in the West. The miner had divided his ores into placer and milling ores, and the latter into free milling and rebellious, by which he meant ores which would or would not readily amalgamate. Such rebellious ores as would not yield to mercury after working, or to which some leaching process was not applicable, were treated for what could be got out of them. To make it possible to treat them by any other process to recover the gold, either copper or lead must be in such quantities that smelting will be remunerative. Such is usually the case to make the treatment for gold alone possible. It occurs in Colorado,

[1] *Trans. Amer. Inst. Min. Engs.*, Feb. 1881.

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where a complicated European process[1] has been introduced with such modifications as were made necessary by local circumstances to treat ores of copper or lead containing gold, to which all the rebellious ores of the district can be profitably added; but the lead is always lost, while the copper is saved as a by-product. The gold thus concentrated in Colorado in a copper product is afterwards separated with little loss. If the gold is concentrated in a lead product, silver must be present, when the gold follows the silver and is afterwards separated from it.

Such methods of smelting are not common. Gold is not usually found in the United States in paying quantities in copper or lead ores, nor have those ores as a general rule been found in any very large quantities in the districts where rebellious gold ores occur, nor if they did, could the separation be undertaken except by men of great skill both in metallurgy and in financiering, as the failure of enterprises of this kind based on either the one or the other, but without both, has shown.

That gold associated with copper, lead, silver, and zinc could be separated by smelting when either lead or copper ore could be had in sufficient quantity, was learned after a long trial, but not until hundreds of thousands of tons of rich tailings from amalgamating processes, containing over four pounds of mercury to the ton in addition to large quantities of gold and silver, had been allowed to run to waste in the streams which flow down the cañons of the mining districts.

We cannot say that we have reached the utmost limit of metallurgical progress in the economical separation of the precious metals, for it has been ascertained that in the hydraulic mines, from which by far the larger part of the gold of California is produced, notwithstanding the great cost of their plant, not more than thirty-three per cent of the total amount of gold contained is saved.

The observer who takes the pains to calculate the millions of dollars contained in the tails which have been allowed to flow away looks aghast at these statistics, but the miner when he comes to consider them makes the very intelligent reply that while he gets out only thirty-three per cent he makes a profit for himself and his stockholders, but when he endeavors to save any portion of the other two thirds he makes either less profit

[1] *Trans. Amer. Inst. Min. Engs.*, vol. 4, pp. 285, 295.

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or none at all, according to the amount of increase which he attempts to make. It is a matter of great regret to him that he cannot save the sixty-six per cent, but his regret is not that it is wealth lost which would be added to the wealth of the State, but that he cannot have it at the bottom of his own pocket; so he abandons it philosophically to the future miner with the expectation that nature will some time concentrate these tailings, or that some process will be invented by which the vast value contained in the tailings that are now lying piled up in many of the cañons of the West can be utilized. In the mean time the filling up of the river-beds[1] and the covering of some of the agricultural lands have become a serious political question, and it may be that in the near future California, which would probably not have been settled but by a mining population, will have to decide whether it will allow the agriculturist, who does not add nearly so much to the wealth of the State, to drive the miner out of the deep placer districts where he settled long before the farmer came there, since the granger element threaten proscription in the shape of legislation which compels the miner to retain his tailings in the side cañons and not to allow them to escape into the rivers or over the arable lands. This question has recently come into the courts of California by an injunction restraining the Miocene Mining Company of Butte Co. in that State, who own 1500 acres on Feather River of placers thirty feet deep, from discharging their tails into this river. The magnitude of the interests involved is shown by the following figures taken from the San Francisco *Alta* of June, 1881:

Miles of ditches and canals in the counties affected..	6,000
Number of men employed, from..... .....5,000 to	10,000
" " Chinamen	500
Capital invested in the mines.....	$150,000,000
Yearly product..... $12,000,000 to	$15,000,000
Mining population affected by the suit.....	130,000
Agricultural " " " " "	60,000

[1] The Tuolumne River in California was formerly five hundred feet wide and fourteen feet deep in the bed of the river, and had a fall of eighteen feet to the mile. In twenty-one months it was filled so that the water hardly ran over the river-bed. In 1875–6 the freshets washed the river clear, but since then it has been again filled, and is now only thirty-one feet wide and one foot deep. The Feather River fills up thirty feet in a single flood from these deposits, but scours itself free again, while the Yuba River at Marysville has filled permanently thirty feet, and fifteen miles above, one hundred and fifteen feet in thirty years.—*Engineering*, London, vol. 25, p. 20.

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It is estimated that should this injunction prevail the amount of farming land now under cultivation which will be rendered valueless by the removal of the mining population will be equal in money value to the total amount of the value of the mining property involved. Should this injunction be made permanent, the State will probably learn too late that a valuable industry has been destroyed by people who found the mines working when they settled on the lands they occupy, and must have known beforehand what was in store for them in the near future. The industry so destroyed it will be difficult to re-establish, and it is very doubtful whether the State will be a gainer by it.

TREATMENT OF SILVER ORES.

As gold grew scarcer silver ores were looked for and became an object of great interest. At first only the rich outcrops of the free-milling lodes were worked, and the ores from them were treated by the old Patio processes which were found in Mexico, in which arastras and Chilian mills were used. Occasionally the Caso method was adopted by some person who had seen it at work or had heard of its working in Chili. The Caso method, working quicker than the Patio, was adopted in some places; the bottom of the box was replaced by iron, and then the sides, and then the idea of grinding suggested itself, until the amalgamation pan in all its varieties grew up little by little. It was at first thought that the pan could be used equally well for both grinding and amalgamating, and some persons still use it for both purposes. But it has been pretty well settled of late years that the machine can do only one kind of work well, and that any other kind of work forced from it is done at the expense of the yield.

The Freiberg barrel did not meet with much favor in the West. Most of the emigrants of 1848 were of Anglo-Saxon origin, and when the German came with his slow but certain methods, a few mills adopted the barrel. But in the early days quick returns were demanded. The pan was an American invention. It moreover had grown on "the coast,"[1] and every one knew or thought he knew how to use it. Every machine-shop was ready to make it, and was more or less interested in one of the many patents taken out for a new one. It gave a return in from three

[1] All the Sierra Nevada district is called "the coast" in the West.

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to five hours, while the barrel worked twenty-four before the amalgam was extracted. The pan required nothing but the ore, mercury, "the chemicals," and water. These had to be used in the barrels, and in addition some iron. Balls were used at first as in Europe, then pieces cut from stamp stems, and then mule and horse shoes, and finally anything made of iron whether adapted by its shape to the purpose or not. The result was imperfect, partly from the want of material adapted to the purpose, and partly because it was not always possible to get the material wanted in a given time. The consequence was that most of the mills that put up barrels found no one to run them, or were obliged to keep one set of men to run and another to repair them. Yankee wit did improve them so as to make each barrel independent by using friction clutches, and by doing away with the cogging by the substitution of friction gear; but a change in the administration usually threw the barrels out and put in the pans, so that the barrel, while everywhere condemned in favor of the pan, has really not been fairly tested. It is cheaper to erect and cheaper to run, but slower in action, and with friction gear ought to compete with the pan.

(*To be continued.*)

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THE MARQUETTE IRON REGION.—II.

BY THE STUDENTS[1] OF THE SUMMER SCHOOL OF PRACTICAL MINING, LAKE SUPERIOR, 1881.

CHARACTER OF THE ORE-BODIES.—(*Continued.*)

Lake Superior Mine.—The stratagraphical structure of the mine is exceedingly complicated, the main deposit apparently being a synclinal basin, the limiting sides of which form the foot-walls of the north and south veins, the former having a dip of about 70° to the south, the latter sloping at a like angle to the north. Between these walls the formation is bent and corrugated into numerous minor folds, all of which have an approximate east and west trend. The strike of the ore-deposit is a little north of west, with a pitch to the west. A geological cross-section taken west of the No. 2 shaft—the main opening into the mine—would seem to indicate that the various folds of which the deposit is constituted assume in general the form of a W, the northern arm of which makes still another fold, upon dipping to the north, and forming the ore-basin which underlies the new Barnum workings, situated on the north half of the northwest quarter of section ten. The facts of the case most certainly lead to the conclusion that the north and south veins, measuring six hundred feet apart along the section-line, will ultimately form a junction beneath the intervening mass, and in that case cut the ore entirely out at that point. Indeed, the present workings on the 320-feet level, the lowest level in the mine, already evidence the truth of this theory, in that, at that depth, they are separated by only a few feet of rock.

. . . . . .

In addition to these two veins—the one lying next to the north wall, the other adjacent to the south foot-wall—there are other minor folds forming a series of undulations, and finally dipping to the north and underlying the swamp immediately to the north of the city of Ishpeming. The general form of the ore masses is lenticular, having a maximum width at the centre

[1] Being extracts from the memoirs required of each student of the Summer School.

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and gradually tapering down, and finally terminating at the east and west ends in the form of a wedge, the extreme edges of which are unprofitable to work. The regular succession of these lens-shaped deposits form, as a whole, a regular vein, having a strike a little north of west and pitching to the northwest. The three middle veins of the formation are characterized by the presence of these "pods," which most frequently overlap one another, though following no apparent law as to which side the overlapping occurs.

In mining a vein of this description, the stope is advanced so long as it continues to yield a profitable grade of ore. A test-hole is then bored in one or both of the side-walls of the vein, to locate the position of the succeeding lens. If the characteristic black color of the sludge from the drill-hole determines the presence of the ore-body, a drift is driven through the intervening rock, and the next lens mined as before, leaving a pillar at the point of drifting.

The north and south lodes appear to be more vein-like in character, the south vein being especially free from any well-defined lentiform masses.

. . . . . .

The rock-formation consists of beds of quartzyte and dioryte underlying the north and south veins respectively, each bed being separated from the ore proper by an intervening layer of magnesian schist, varying from a few inches to several feet in thickness. This schist, though resembling in composition that of other portions of the Marquette region, has certain distinctive characteristics which serve to distinguish it *en masse* from that of other localities. It is soft, compact, crystalline, and made up chiefly of feldspar of a light green or greenish black color, and of chlorite scales, together with more or less magnetite in small crystals. Its cleavage is marked, its planes inclining at a large angle, having the general east and west trend of the formation. The intermediate layers, lying between the upturned edges of the basin, consist of alternate beds of specular ore and schistose rock—the former, as frequently the latter, being arranged in lenticular-shaped masses having a maximum thickness at the centre of many feet.

The ore itself belongs to that class known as specular hematite, or red specular ore, consisting essentially of iron oxide with a few per cent. of silica and minute quantities of phos-

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phorus. The yield of metallic iron averages about 64 per cent., the amount of silica being between 3.5 per cent. and 4 per cent.; phosphorus about .04 per cent.

The mine produces two varieties of specular ore. The one readily splits into slabs, shows a bright surface when freshly fractured, and is known as specular slate-ore. The other variety, a granular specular, is massive, compact, exceedingly hard, and shows no tendency to split into slabs. A certain variety of this latter ore constitutes what is known as "fine-grained steely ore," a vein of which occurs in No. 2 Shaft at the two hundred and forty feet level. It is of the hardest and most compact nature, contains little silicon, yields a high per cent. of metallic iron, but is also high in phosphorus, and therefore undesirable for conversion into Bessemer pig.

The above refers only to first-class ore, which amounts to about eighty-four per cent. of the entire ore taken from the mine, the remaining sixteen per cent. consisting of a mixture of the above with a reddish ferruginous quartz, the whole constituting the second-class ores. This quartz-constituent, which proves itself so deleterious an ingredient in very nearly all the formations of this region, is jasper, and it occurs associated with the ores in alternating laminæ of varying thickness, usually showing conformable stratification, but occasionally bending and breaking up into the most contorted shapes, and obliterating all evidences of its true bedding-planes. By a thinning out of the jasper layers the mixed variety merges into a first-class ore. More than twenty per cent. of quartz in any ore condemns it as worthless for metallurgical purposes, though such a mixture has always played an important part in the discovery of many of the present Lake Superior mines; for while the pure ores yield readily to erosive action, and, becoming covered with soil, give no visual indication of their presence, those containing intermingled jasper-bands are not easily affected by weathering, and, as a result of this increased hardness, offer by their outcrops tangible proof of the existence of ore in the vicinity.

W. A. SHUMWAY.
D. B. TOUCEY.

The New York Mine.—The forty-acre tract on which the New York Mine is located is crossed apparently by three veins of ore, all having a nearly east and west direction at the

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surface, and dipping from 30°–45° to the south. Careful geological surveys seem to indicate that the two more southerly veins are in reality one, that they were formerly connected by a sharp anticlinal curve, and that erosion of the upper portion has left two apparently distinct parallel veins separated by a bar of rock fifty feet thick.

The upper or southerly half of the fold has at the surface a westerly direction, which changes a little to the north in Pits 2 and 3, and regains its westerly direction in Pit 1.

The northern half of the fold has at first a more northerly direction than the other, but turns near the east end and becomes nearly parallel to the first. The third vein does not appear to be connected with the others, and was probably an earlier deposit. It has a dip of nearly 60° where now worked, and is thought to be leading to extensive deposits.

Too little exploring has been done to form any complete idea of the geological structure. The ore is a very rich hematite, always hard, but with degrees of hardness. In Pit 1 a machine will at times not make more than five or six inches in ten hours, and spots have been struck so hard that the mine blacksmith could not temper steel to cut through them. The ore from the third vein is exceedingly pure, worth two dollars per ton more than any other.

The hanging is in nearly all cases quartzite, but in Pit 3 the quartzite pinched out and was replaced by soapstone. The quartzite hanging is very stable and needs no layer of ore left to support it, and but few pillars.

Pyrite is found in isolated spots, and also a black, micaceous mineral, which is probably only hematite ; quartz and jasper are also mixed with the ore in small amount.

A. J. MOSES.
W. F. STAUNTON, JR.

The Cleveland Mine.—The deposit which is at present most worked and apparently the largest is the Incline. The foot-wall of the deposit dips at first to the northwest, but in the southern part of the pit changes to about an equal dip in the opposite direction. The hanging wall is mostly chlorite and the foot-wall jasper, but this is sometimes reversed, and sometimes both hanging and foot-walls are banded jasper.

The thickness of the deposit varies from forty to sixty feet, the average being nearer forty.

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The ore is all hard specular hematite, but varies very much in different parts of the pit.

In many parts of this and Sellwood's Pit the slips are filled with soft, white material, feeling like talc—probably kaolin.

The chlorite is quite soft, but tough, and when not too flaky, as it usually is, makes a safe hanging wall.

The probable extent of the deposit may be judged when the fact is considered that apparently the same deposit of ore, still having the same thickness, is struck by a vertical diamond drill-hole at a depth of 326 feet. It also becomes almost a settled fact when we remember that the deposit called "Sellwood's Pit" dips toward the southwest, that both these deposits are united at some point between them and west of each.

In Sellwood's Pit, the second in importance, the dip is toward the southwest in a general way, but the foot-wall, where it is visible, is quite irregular. This deposit is the same as that of the New York Mine; it is supposed to form the northern edge of the great Ishpeming ore-basin. The hanging wall is generally chlorite and the foot-wall jasper.

In the Sawmill Pit, the deposit extends southwest and northeast. The hanging-wall appears to dip to the southwest, while the foot-wall, though dipping in the same direction, is more nearly horizontal, and it seems likely that the deposit may be cut off toward the southwest, by the meeting of the foot- and hanging-walls. Swineford regards the deposit as connected with that of the Incline, but the miners consider it simply a large pocket.

W. S. PAGE.
W. S. WHITE.

The Jackson Mine.—The hard (specular) ore generally occurs between jasper and jasper, or jasper and soapstone.

The surrounding rock is, in general, a mixed ore, called "jasper," consisting of laminæ of a red jasper, alternating with ore in variable ratio—in many cases containing so much good ore as to make it quite valuable in any less favored district; and even here, in some mines, the best qualities are picked and sold as second-class ore.

The folds in the country rock are so great and complicated in the vicinity that a study of the stratification is made quite difficult. At the ore-masses the strata are so nearly vertical

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that it is in places difficult or impossible to tell at sight which is hanging and which is foot wall. In such case, there is little danger of falling in or caving of walls, and but little supporting timber is needed. The jasper, which is so abundant in this mine, is a strong, tough rock, which fact is another excuse for using little timber.

The greenstones, quartzites, traps, etc., seem to be absent or in very small quantities and with little significance, and even the soap-rock and jasper seem to show but little system in their occurrence, as they are about equally divided between the foot and hanging walls.

A careful study, however, of the maps, and of the ore masses in their mutual relations, shows that this irregularity is more apparent than real, and that in all probability it is due to the very great folding of quite regular strata; and that originally these ore masses were deposited on the jasper by some action of water, and over and between them layers of "soap rock" were deposited. This soap rock is the equivalent of the chlorite of many of the mines of the western part of the country, and is in most places quite thin; but in a few cases, as in the great horse in Pit No. 5 and north of the main shaft at the tunnel level, it occurs in large masses. This peculiarity of a continuous coat of sedimentary rock over the bodies of ore, and the occurrence of masses of the same cutting out the ore in some places, reminds one of the shale of the coal measures, and the "horses" of the same, reaching into the coal just below; and it is probable that the two are formed in the same way, the horses being simply the sedimentary casts of water-worn cavities and furrows in or between the ore-beds.

Other rocks occur in small quantities only. "Magnesia," probably a very light talc, is common, in pieces of small size, in the mixed ore and rock at the extreme end of a lens, and it is consequently a bad sign.

There are one or two veins of quartzite found in the exploring works, but they seem to have no import, as they are in all cases far from ore, and are of small size.

In some parts of the mine there are masses of "rotten jasper," so-called; in reality, a friable ore with considerable impurity, and resembling a stage between specular ore and hematite. This is generally too impure for use. In some few places a similar decomposition seems to have taken place in the pure ore, and the resulting hematite, although not so de-

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sirable as the specular, is mined in small quantities. This hematite occurs only in small amounts.

"Greenstone" occurs in one or two places only, and then in small quantity.

"The ore" is a hard, fine-grained specular, not nearly so micaceous as that of the "Republic Mine," and much less brittle. It is similar in general character to the usual hard ores of the Negaunee District, but in some places is so rich in manganese as to be specially valuable. Some specimens run as high as forty per cent., and a whole series of cars have been loaded with ore running from ten to twenty per cent. manganese. This specular ore, as is well known, is really a true hematite, and it is very red when powdered; for this reason the whole mine, and all the roads, men, tools, etc., are brilliant examples of hematite "streak."

The majority of the lenses of ore lie in nearly parallel positions, and are arranged in a line, end to end, with masses of rock between and in them. In the latter case the masses are called "horses," and they are very curious, often being of great size and still entirely enclosed by ore, which also cuts them out at the bottom. For this reason they are very troublesome, and their great size necessitates large pillars of ore and heavy timbering to keep them in place.

In most cases these "horses" are of jasper, and they are generally placed very close to the "foot-wall"—even, in some cases, so near that the "stringer" of good ore which separates them from the foot-wall, although always present, is of unworkable thinness.

The strata right at the mines seem to be without faults of notable size; but their folds are so very great and complex, that they are as bad as faults, or worse, and give the mine a very irregular appearance.

The "lenses" vary greatly in size, from those several hundred feet in length and fully one hundred feet wide, as seen in some of the old workings, to simple "chimneys" of considerable height, but of small horizontal cross-section. Also, on the surface, there are, or rather were, masses of ore which, when worked, ran out very soon, and seem undoubtedly to be the lower parts of lenses, in some cases probably very large, of which the upper parts have been removed by glacial and other erosion.

F. B. CROCKER.
J. B. PORTER.

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EARLY METHODS OF MINING.

UP to 1870 the system of mining employed was that of open excavation, a system which, considering the undeveloped condition of the country at that time, and the too ambitious haste of American capital for interest on investments, is entirely defensible. The method of sinking shafts, running drifts and cross-cuts, driving levels, etc., which the canons of a more perfected system of mining demand, involving, as it does, a larger preliminary outlay, had but little support, and no doubt, the conditions remaining the same, the same principles of mining would have been in operation for long afterward.

The constantly increasing depth of the mines, the increased cost of handling the product, the large amount of waste, the introduction of powerful and elaborate machinery, the requisite enlargement of the plant, absorbing a large portion of the company's funds, have, however, made the employment of the most advanced mining engineering skill an absolute necessity. At the present time, when a scientific study of the nature and method of the deposits is being pursued, the importance of underground plans is thoroughly understood, while to systematic working and efficient management they are of vital consequence.

The earlier workings on the Lake Superior, and indeed on most of the Marquette mines, differed but little from ordinary rock excavations. The most effectual method, and the one generally employed, was the so-called "sand-blast." Holes of 10 or 15 feet burden were drilled and preliminary charges fired, shaking the ground and producing cracks and cavities at the bottom of the hole. The cavities becoming enlarged by the repeated blasts, a final charge, consisting of one hundred or more pounds of powder, was introduced, and the crevices filled with sand. The entire charge was fired at once, the material, amounting to many hundred tons, being thrown over into the open space previously excavated.

PRESENT METHODS OF MINING.

At the Michigamme Mine, after the open pits had been worked out, the following system of mining was introduced:

Twenty-five feet below the adit level a drift was driven, 2,000 feet long, the length of the proposed workings, and through the

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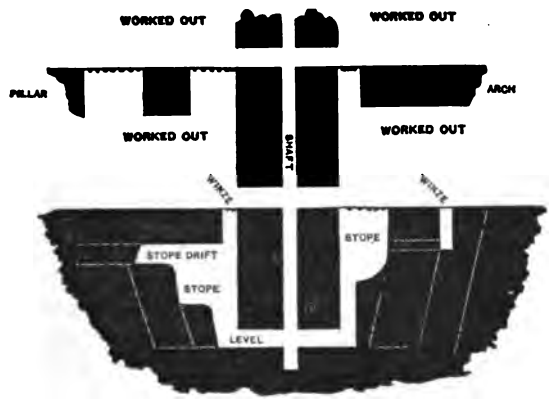
arch thus left shafts were sunk at intervals to a depth of 100 feet below the adit level. Below this the levels are uniformly 50 feet apart, and the arch left 15 feet thick. The shafts are all 8 feet long by 14 feet, and are sunk in the ore next the hanging-wall. On each side of the shafts, and 15 feet distant, winzes 8 feet by 8 feet are sunk, thus leaving 15-feet walls as a protection for the shaft and support for the roof. The winzes connect with the shaft at the levels by drifts 8 feet by 8 feet in cross section. The ground is broken by underhand stoping, which is considered the most economical way, and dropped through the winze to the level below, whence it is trammed to the shaft and hoisted to the surface. The ore is thus mined out laterally, as shown in the dotted lines on the right-hand



Sketch showing method of stoping in the Michigamme, Champion, and other mines.

side of the figure, to a distance of fifty feet from the shaft, when, by drifting and sinking a special winze, a pillar of ore is left to support the hanging-wall, and the stoping continued below and beyond the pillar, as shown. The sizes of pillars left and the distance between them are slightly varied, according to the character of the ground. They are generally so arranged as to break joints from level to level, and stulls are frequently placed between them for additional security. Where the hanging-wall is bad pillars are more frequent, and do not break joint; in one case they are continuous from the 4th to the 7th level.

In this mine it is the custom to do all the dead work, *i.e.*, drifting, sinking, opening up, etc., in the winter, thus saving expense of storing, rehandling, etc.

At the Champion Mine, also, the underhand method of

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stoping, or a modification of it, is employed. The lenses of ore are from 20 feet to 40 feet wide, have a very steep inclination or dip, and great care has to be exercised to leave sufficient ground unexcavated as supports for the shafts and levels. Walls of ore are therefore left on each side of the shafts from level to level. The first thing in the removal of the ore is to sink a winze, at a distance of 15 feet to 20 feet from the shaft, down to the level below. If a floor-arch must be left for the support of the track, a drift is run from the winze, 15 feet or 16 feet below the level, and is called the "stope drift," as shown on the left-hand side of the figure on page 111. It is always kept sufficiently in advance of the stope to afford room for setting up a machine drill. Holes are now drilled 10 feet to 12 feet deep, and blasted with nitro-glycerine; the ore being thrown down the winze, broken up by sledging, and trammed to the nearest skip shaft. Nitro-glycerine is used only for stoping, in this mine, and it is found to be most economical to bore a single deep hole and put in a heavy charge, several hundred tons of ore being frequently thrown down at one explosion.

It is customary to pay the miners and trammers 80 cents and 50 cents respectively per skip of ore hoisted.

At the Republic Mine, on account of the great size of some of the ore bodies, in one case 160 feet by 430 feet at the lower level, the mines cannot always be worked by the ordinary method of underhand stoping. A system of chamber-working was therefore adopted for these large ore-bodies, in which the stopes run across the deposit from foot to hanging-wall. To make the shaft secure, an enormous block of ore has been left, as shown in the sketch. Levels are driven out from the shaft through the shaft pillar; at the same time, winzes have been sunk in the chambers on either side. When the shaft and winzes are finished and connected by drifts at the new level, underhand stoping is commenced, the ore being blasted out, commencing at the top of the winze. A large chamber is thus gradually formed of a length equal to the breadth of the vein, about 30 feet wide, and from 50 feet to 150 feet high, according to the character of the ore.

In connection with this method of chamber-working may be noted the system of filling rooms with waste rock, as employed in the Lake Superior Hematite Mine.

The dangerous character of the ground in which the ore is

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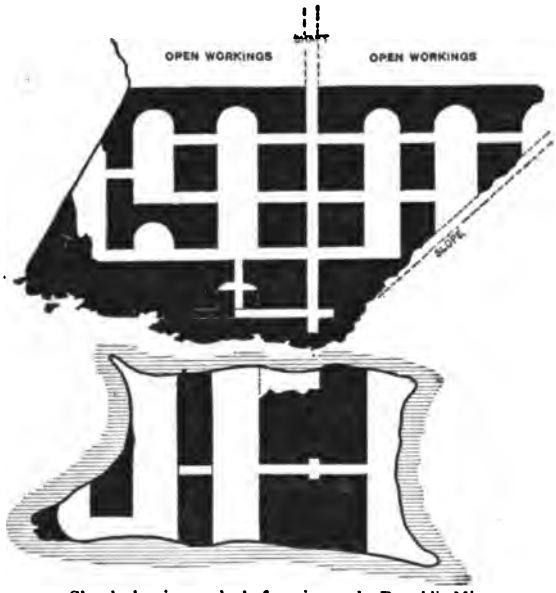
situated, necessitates some system of roof-support different from that generally followed. Timbering would not do, there being no foundation for timbers to rest upon, so the idea was conceived of mining the ore in chambers running from foot to hanging-wall, leaving an arch of about 30 feet thick on top, and having the rooms average 45 feet in length, along line of vein, pillars of the same dimensions as rooms being left between.

The dimensions of the rooms vary on different levels according to the judgment of the mine-captain, but care is always taken that the pillars of one level shall rest on those of the level below. Pillars are cut by breast-stoping on both



Sketch showing method of stoping at the Republic Mine.

sides of the tramway to the height of 30 feet, when overhand stoping is begun, the miners being protected meanwhile by a system of timbering called spiling or fore-poling.

To describe the method of filling, let us, leaving the pillar around the shaft, commence mining in room 1, west level 1. Commencing at the sides of the drift, we breast-stope to the foot-wall on one side and to the hanging-wall on the other, going along the drift a distance equal to the length which we desire the room to have. Breast- and overhand-stoping goes on until the arch of the room has been formed, the underside of the arch being like a segment of a sphere, in which the highest

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part is 30 feet above the floor. The drift is now timbered in with "sets," consisting of legs and cap-pieces set about 4 feet apart and covered over on top and sides with heavy (variable) 12-inch cedar lagging, a mill 5 × 5 feet being built alongside.

The miners now spread all waste rock mined over the floor of the room until it is high enough to give them a support while working, sending the good rock down the mills; and should there not be enough poor rock in the room, some is carried in from other parts of the mine and dumped where desired. The overhand stoping is carried on until the rooms average 70 feet in height, the filling having till then been done by miners in the room; and to complete the filling a winze is cut from the level above and rock sent in through a mill from the surface until the room is entirely filled. There are two such surface mills.

When the New York Mine was first opened, it was worked as an open cut; but as the dip of the deposit was considerable, the underground method soon became necessary, and the system was adopted of cutting out chambers along the face of the deposit, leaving pillars of ore at intervals to support the hanging wall. These pillars were weakened by the fall of similar workings in the Cleveland Mine adjoining, and giving way, caused an extensive cave-in. The mine is now worked in pits, the plan followed being to sink a shaft in the vein and as soon as possible beyond the entrance, to drift to the right and left and stope down. The width of the pit is generally decided by some breast of rock crossing the vein, or by the character of the ore. The work is very little more than quarrying; a few pillars are left at irregular intervals, but generally the roof is sufficiently strong to support itself—the miners simply excavating a pit and sending the ore to the surface.

The system of ore-extraction employed at the Lake Superior Mine is that of sinking shafts on the deposit, and at distances of forty or sixty feet, opening levels in the vein, along which the ore is mined out by a series of underhand stopes, the dimensions of which are dependent entirely upon the character of the ore and the nature of the deposit. In the arrangement of the stopes, etc., the method is similar to that described as used in the Champion and Michigamme mines. Access is further obtained to the different parts of the mine by subordinate shafts, sunk from one level to the other for the purposes of ven-

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tilation, and laying open sections of the vein to the best advantage for stoping them away. The material taken from the stopes is broken, sorted, loaded into cars, and trammed to the shaft, at the mouth of which it is delivered by skips operated by the engine and drums in the engine-house.

SHAFT-SINKING.

The position of the shaft is in any case determined by the following considerations:

First.—The situation and underlie of the deposit.

Second.—The nature and distribution of the ore-bodies, the amount of barren gangue, etc.

Third.—The character of the wall-rock, the ground most favorable for sinking, providing it does not interfere with other and more important considerations, naturally being selected as involving less expense.

Fourth.—The point giving the strongest indication of productive working.

Fifth.—The value of the upper portion of the deposit, the sinking of a shaft through the ore necessitating the leaving of a pillar of ore on each side to prevent it from falling in.

In sinking a shaft to a lower level, it is customary, from economic considerations, to continue it some twelve or fourteen feet beyond the level to be opened. This method at the same time allows of ready access to further sinking, and does not in the least interfere with the raising or lowering of the skip in case it is desired to open a still lower level.

Two new vertical shafts have recently been sunk to reach a deep deposit of ore on the Barnum Mine property. Shaft A has a cross-section of 16 × 12 feet, and has now (July 17, 1881) been sunk 395 feet, principally through quartzite.

The sinking was at first by hand-drilling, but since March 1st, 1881, machine drills have been used. The following table gives a comparison of the two methods:

Average number of holes drilled per day by hand.....	11.2
" " " " " " machine..	14.5
" depth drilled per day by hand.....	31.3
" " " " " machine	47.7

The sides of the shaft were squared up after each blast, and every 16 feet the bottom was cleared out and a new set of timbers put in position. In timbering, wall-plates were first set

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in place, their ends resting in notches, ½ to 1 foot deep, cut in the wall, and being held in place by wooden wedges. Upon these wall-plates the entire 16 feet section of close-set 12-inch square pine timber rests, and in the 3 or 4 feet of clear space below the sinking is continued.

While sinking through the sand of the first 40 feet, two-inch planks were driven in back of the heavy timber to keep out the sand and water.

The shaft is divided by 12-inch square timbers into two parts, the "down cast" and the "up cast," the former for hoisting and lowering; the latter contains the ladder-way, pumps, and steam- and air-pipes. Every 16 feet three 12 × 12-inch timbers are placed across the up cast, serving at first as platforms for the sinker pump, while later the alternate ones serve as ladder platforms. The ladders are 33 feet long, made of green hickory, and fastened to the division timbers by 6-inch staples.

At a depth of 44 feet water came into the shaft at a rate of 250 gallons per minute. After sinking 25 feet in the quartzite ledge, below the sand, a clay seam 6 inches thick was encountered, which led in the water from the surrounding land in enormous quantities.

A drift was now driven on one side of the shaft, and a catch-basin, 21 × 14 feet, and holding 9,000 gallons, cut in the rock to receive the inflowing water. The water was directed into this basin by a gutter around the inside of the shaft, made simply by fastening bevelled planking to the inside timbering. The spaces between the rock and gutter were filled with impermeable puddled clay.

A No. 10 Knowles pump is stationed at this point of the shaft to force the water from the catch-basin to an adit which meets the shaft 30 feet below its mouth, and through this adit it is carried out by a launder at an incline of about 1 to 100.

It was then found that a No. 5 Knowles pump, by following the sinkers down, could keep the shaft free from water during the sinking. This pump was shifted 16 feet at a time, as the sinking progressed. At 130 feet below the first catch-basin it was found necessary to cut another, holding 600 gallons, and to set up another No. 5 Knowles pump, as this distance proved to be the limit of the forcing power of the pump.

The description of the sinking of Shaft B, of the Barnum Mine, through quicksand, will be given in another place.

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The shaft of the Lake Superior Hematite Mine, having a cross section of 8 × 12 feet, was sunk in lengths of 5 feet or so at a time, and timbered as follows: Two 12 × 12-inch pine timbers were placed against the 12-foot sides, the ends being cut so that two shorter pieces of same cross section could be fitted on, making what is termed a toggle-jointed frame. A brace made of 12 × 12-inch timber was now fastened across the middle of the frame, or "set," as it is called, and then a sheathing of planks 1½ × 12 inches × 5 feet were driven in around the set and against the natural walls of the shaft until the heads of the planks were only a little above the set. Another set was put in 4 feet below the first in the same manner, driving the sheathing over the first sheathing. Four posts, 12 × 12 inches, were now put between the sets 4 feet apart, at each of the four corners, the shaft excavated further, and the same method continued.

DRIFTING AT THE LAKE SUPERIOR HEMATITE MINE.

There are two ways of proceeding, according as the face of the drift is composed of hard or soft material. In the first case, suppose the face of the drift to be divided by four horizontal planes, a line of blast holes, in depths ranging from 4 to 8 feet, is drilled in the top of the first or bottom division, these holes slanting downward, so that when charged the whole wedge of the lowest division must break out to the depth of the holes. Doing the same for the next division, it also comes out, and so on until the entire face is removed to the depth of the holes.

When drifting through softer material, as the ore, for example, the method was simpler. A line of holes was drilled in one side of the face, slanting toward the centre, another line of holes at the bottom, slanting upward, and one hole in the upper diagonal corner of the face, slanting downward. When fired simultaneously they brought down the whole face in the form of a wedge.

(To be continued.)

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RECLAMATION OF TIDE-LAND AT MARSHFIELD, MASS.

By WILLIAM HILL, '82.

THE reclamation of tide-land, though practised to a limited extent ever since the first settlement of this country, has not been of great importance, owing probably to the low price at which the neighboring uplands could be obtained. As the sea-coast becomes more densely populated, and the demand for land more urgent, the reclamation of marsh-land will be looked upon with more favor and interest than in the past. In this country the utilization of tide-lands has been limited to small areas, reclaimed for special agricultural or building purposes. The reclamation of these lands has been accomplished by dyking to prevent the influx of salt water, and ditching as a means of drainage.

The marsh-land reclaimed at Marshfield, Mass., has an area of 1,412 acres, of which about 200 acres at the inland end were nearly fresh previous to dyking. The marsh is divided by a small stream passing through it. About 752 acres are situated north of this stream, and 660 are south of it. The surface of the marsh was about on the level of high-water mark, so that during the low course of tides it was but slightly overflowed, while on the spring tides it was flooded to a depth of from one to three feet. The soil was black and spongy, too soft in most places to bear up a team.

Salt hay of poor quality was its only product.

The water-course through the marsh, and known as Green Harbor River, extends about three miles back into the country, and receives numerous small tributaries, making it in spring a stream of some magnitude, but in a dry time the flow of fresh water is very small. The lower end of the river, below the site of the present dyke, is known as Green Harbor. The entrance to this harbor was obstructed by a sand-bar, having at low-tide about two feet of water upon it. The mean rise of tide is about nine feet, giving, at high water, a depth sufficient for large craft to cross the bar, and once inside there was ample water at all stages of tide for small boats to anchor in safety.

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In 1871 the principal owners of the marsh land obtained from the Legislature an act granting them the right to construct a dyke across Green Harbor River "at or near and not above Turkey Point."

The act also provided that should shoaling take place in the harbor, above mean low-water, it should be removed by the owners of the marsh, or in case of their failure to do so in six months after notice from the Harbor Commissioners, it might be removed by them at the expense of the marsh owners. The money necessary to build the dyke was to be raised by assessment upon the marsh-land. Though some owners opposed the reclamation, it was decided to build the dyke between Turkey Point and the north side, at a distance of 3,300 feet from the mouth of the harbor.

The dyke was to be six and a half feet high above the marsh, and extend from Turkey Point on the south side of the river, 1,800 feet in a course north six degrees east, to the north side.

In its course the dyke crossed three channels, viz., one between a small island and the south shore, the main channel, and a small water-way between a larger island and the north shore. First, two sluice-ways, for the discharge of fresh water from the marsh, were built between the small island and the south shore. Three rows of 4-inch sheet-piling were driven across the channel, one row under each end of the sills of the sluice-ways, and one row midway between. Except where the sluice-ways passed through them, these sheet-piles extended to high-water mark. The sluices were side by side, 4 x 4 feet cross section inside. The bottoms were at extreme low-water mark, or about 17 feet below the top of the dyke. The sluices were framed of 8-inch timber and planked longitudinally on the top and bottom.

Each sluice-way is provided with gates, which swing automatically about a vertical axis, the hinges being of gun-metal.

These gates open toward the sea, and remain open while the level of the water outside is below that of the water in the marsh, but when the tide rises so that the current would be inward, it closes the gates, hence fresh water can flow out from the marsh at every low tide, but salt water can not flow in, the gates preventing an inward current.

Across the marsh a ditch was dug on the centre line of the dyke, down to hard clay, so that when the marsh settled under

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the weight of the dyke, it should settle evenly, the marsh mud being pressed out from the ditch on each side. This would cause the surface of the marsh under the dyke to assume a V shape. A common earth-work was then built out on each side from the upland, the material used being gravel found in the vicinity. The top of the dyke was six inches higher than the highest point reached by the tide during the "Minots Gale" of April, 1851, when it reached the highest level ever known.

A slope of 1½ horizontal to 1 perpendicular was given to the inside, while the outside was in three different slopes, viz.: 1½ to 1, 2 to 1, and 3 to 1, the steepest part being at the top.

The small channel on the north was filled by a common dump, in the other small channel was the sluice-way, and the third and largest was the site of the tide-way or apron. This apron was near the south side of the main channel.

It was designed to give a passage for the tide until the dyke was in a condition to exclude the water from the marsh. Between the apron and the north shore of the channel two parallel rows of piling and sheet-piling were driven, and connected by iron tie-bars.

The sheet-piling reached to high-water mark. Earth was then filled in between these two rows of piles.

The apron was 70 feet long, measured across the channel.

For its construction, two rows of sheet-piles, 18 feet apart, were driven about 10 feet into the mud, when they struck hard-pan which they could not penetrate.

These sheet-piles were cut off at low-water mark.

The space between the two rows of sheet-piles was decked over on low-water level. On top of the deck, on its middle line, a row of vertical timbers, five feet apart, was set up. These vertical posts were double, and so bolted together as to leave a space of two inches between them, and were braced by braces from their tops to the edges of the apron. The five-foot spaces between the posts were arranged to receive flash-boards slid down in the grooves.

When in position the flash-boards would stop the flow of water between the vertical posts and over the apron.

The plan was to build the dyke from each side out to the apron, the water flowing in and out over the deck in the meantime. When the dyke was complete on both sides, the flash-boards were to be placed in position at low tide, and the ocean

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permanently shut out from the marsh. The whole portion in the channel was then to receive a fill and take the proper slopes outside of the buried structure.

The work proceeded well until the dyke south of the tide-way was finished, and the filling of the channel between the north shore and the tide-way or apron was commenced.

As the filling proceeded the water-course was contracted, and a large part of the water was forced through the tide-way, 70 feet wide, provided for that purpose. When that portion of the channel north of the tide-way was nearly filled, the water-course was contracted to such an extent, that the discharge of water from the marsh was very much obstructed, so that the water in the harbor below the dyke fell so much faster than that above, that its surface became six feet below the level of the water above the apron. This state of affairs caused the water from the marsh to flow out over the apron with tremendous force, and to fall over its outer edge as it would over a dam. The action of this current was to cut away the mud and rip-rap from the lower side of the apron, in consequence of which the sheet-piles began to float off down stream.

At this point the situation was indeed critical. All the available teams were summoned to fill in the tide-way with whatever material could be most quickly obtained. Bags of sand had been provided in anticipation of such an accident, and they, with stone walls, gravel, and other similar material, were thrown in to fill the breach. After some time of anxious and hard working, the channel was closed and the structure saved by a rough fill.

It is said that the rush of water was so great that stones of half a ton or more in weight were carried several hundred feet down stream by the current. Had the tide-way been wider, or heavier rip-rap been used to prevent the cutting away of the mud by the current, this disaster would have been avoided.

It augmented the cost of the dyke to a considerable degree, as damages had to be paid for the walls, etc., removed to fill the channel. The extra expense on account of the accident was more than $5,000.

When completed, the structure was four feet wide on the top, and about six and a half feet high above the surface of the marsh. In 1879 it was widened, at a cost of a little less than $3,000, to carry the road between Brant Rock and Green Harbor Village.

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The total width is now about thirty feet. The total cost of the dyke was $32,261.68, the items of expense being as follows:

For land, gravel, and similar materials	$2,435 20
" lumber and piles	2,842 46
" sundry labor	3,941 79
" bolts, rods, spikes, etc.	570 41
" pile-drivers' labor	3,647 70
" gravel filling	12,049 01
" sluices and tide-gates	1,520 64
" sundry expenses	960 84
" interest account	1,131 71
" legal expenses	582 00
" salary of commissioners and collectors' fees	2,579 92
Total cost	$32,261 68

After the completion of the dyke, ditching should have been immediately commenced, but on the contrary it was very much neglected, owing to the lack of funds and the indifference of all the owners, except the most sanguine. A few ditches were dug from time to time, but no thorough system of drainage was attempted. The water level in the stream flowing through the marsh is about five feet below the marsh level, hence the average depth of ditches is about four feet.

It was found that ditches 1 foot wide and 2½ feet deep, cost about twenty-five cents per rod in length.

Previous to the construction of the dyke, the marsh was similar to those found along the coast, producing only a poor quality of salt hay, and vegetation common to such land.

The soil was of a dark, rich color, exceedingly spongy and soft, so that a pole could be run down from eight to fifteen feet before bottom was reached.

When first reclaimed, the soil was of course very salt, being even more so than the ocean itself, probably on account of the continued concentration of the sea-water by evaporation.

The following analyses, made by Professor Goessmann, of the Massachusetts Agricultural College, show the number of parts of chlorine found in samples of the marsh-water, as compared with the amount contained in sea-water. The samples were collected in May, 1873, the salt water having been excluded the November previous.

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Chlorine found in one hundred parts of water, was as follows:

Sample I.	Sample II.	Sample III.
1.9407	1.702	2.3195

Sample No. I., percentage of chlorine in sea-water.

Sample No. II., percentage of chlorine found in a sample of water taken from a hole three or four feet deep, dug for the purpose, in the middle section of the marsh.

Sample No. III., percentage of chlorine found in a sample of water from the lower section of the marsh, collected in the same way as No. II., the sample being somewhat stagnant.

From the foregoing statistics we see the great importance of drainage, in order that this salt may be removed by the infiltration of rain-water, which would carry it off in solution.

While drainage is so insufficient, but poor agricultural results can be expected, as the salt, though dissolved by the rain-water, can not be removed.

The crops raised have been principally hay and onions, though corn, potatoes, oats, pumpkins, and a few strawberries have been tried with success. Onions have produced the most satisfactory results, being less sensitive to salt than many crops. In 1880, seventeen hundred bushels of onions and four hundred bushels of corn were raised. Experiments with hay have shown that red-top produced the best results, though timothy was found almost equally good. In 1880 one acre of red-top produced three tons of hay.

At no time has there been more than three or four hundred acres of marsh land under cultivation, and a considerable portion of it is now covered with birches, some of which are fifteen or twenty feet high.

The salt in the soil is still a hinderance to cultivation, preventing the maturing of crops in some localities, but, with working and drainage, this evil is gradually disappearing, and in due time will probably be entirely overcome. It is not the custom to plow the land, harrowing being all that is necessary, owing to the lightness of the soil. Considerable difficulty has been experienced on account of the old roots of salt grass, which fill the soil to a depth of several feet, having accumulated apparently for centuries without decay. They hold the soil in a peat-like mass, making it difficult to break it up sufficiently to make a fine mould. Continued working has to some extent

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eradicated the difficulty by breaking up the mass, but the greatest relief has come from a fire which burned over about ten acres of marsh, and completely removed the roots to a depth of six or eight inches, leaving a fine soil. All crops can be raised on the marsh land with much less labor than on the uplands, owing to the lightness of the soil, its freedom from stones, and the fact that no fertilizer is necessary.

When these facts are considered, it seems evident that crops can be produced for much less than on the uplands. In one instance potatoes were produced at an outlay of twenty-five cents per bushel.

The former value of the marsh land averaged about $6 an acre, at present it is valued by the owners at an average of $40 an acre, or more than six times its former value. With only this increase in value of $34 per acre, on 1,412 acres we have a return of $48,008 dollars for the money expended in the reclamation.

The value of the marsh land will increase every year as the soil becomes in better condition.

From these considerations it seems that, were it not for legal difficulties, growing out of the shoaling of the harbor by reason of the dyke, the financial aspect of the enterprise would be good.

When the saltness of the soil shall have been overcome by time, proper drainage, and working, and its fibrous condition removed by fire or decay, it will doubtless amply repay the expense incurred, by its ease of working and inexhaustible fertility. While the benefit to the marsh land is great and increasing, the damage to the harbor below, by reason of the dyke, is very marked.

Previous to the construction of the dyke, Green Harbor was about 500 feet wide, and extended from the present position of the dyke 3,300 feet in a southerly direction to the ocean.

The north side of the entrance of the harbor to the ocean is a rocky point, while on the south side a beach stretches away nine miles to Gurnet Head. At the entrance to the harbor was a bar, which still remains.

It was observed that when a northeast storm occurred at a time when the tide did not overflow the marsh, *i.e.*, at the low course of tides, that the sand was driven into the harbor. This sand remained in the harbor till the succeeding high course of tides, when it would be washed out, and the harbor be left in its previous condition. From this fact alone we see

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that the water which overflowed the marsh land played an important part in the scouring of the harbor, as indeed we might suppose it would.

The marsh being at the level of mean high-water, was not covered until toward the last of the flood-tide, and high-water on the marsh did not occur till after high-water in the sea.

When the ebb-tide commenced, the marsh served as a great reservoir, holding back the water upon it, by reason of its large expanse, until the water-level in the harbor below had fallen, and then causing a rapid outflow at a low stage of the tide. The scour of a current is as the square of its velocity. In the case of the flood tide, when it first began to make, *i.e.*, when the channel through which it must run was small, the velocity was also small, because there was but little filling to be done, and when the tide reached a point where it began to overflow the marsh, the channel at the surface was much broader than that occupied by the water at low tide, and consequently the velocity would not be greatly increased.

On the contrary, during the ebb tide, the greatest flow would take place when the tide had fallen sufficiently to allow the water from the marsh to flow out, its velocity becoming greater and greater as the tide outside fell, giving a greater head and a narrower channel. It is evident, therefore, that the greatest velocity of current was attained near the last of the ebb tide, and the greatest movement of drifting sand would then take place.

It would seem that an equilibrium existed between the forces which tended to drive the sand into the harbor and the scour which tended to wash it out. The bar was formed on the neutral ground between the two forces.

If one of these forces was diminished, the other must gain the ascendancy and control the changes till, by alteration in the channel, equilibrium was again established.

This was the case when a storm occurred on the low course of tides, for then the efflux, being small, was impotent to force back the sand driven in by the waves, and it remained until the marsh was again overflowed, when it was washed out. When the dyke was built it shut off the water from the marsh at all stages of the tide, and the same result followed as though the low course of tides had been permanent. The consequence was a filling-in of the harbor.

The filling commenced near the mouth and has worked

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farther up each succeeding year, though not as rapidly latterly as for the first few years after the dyke was built.

The decrease in the rapidity with which the harbor has filled in of late is probably owing to the decrease in size of its mouth, giving the sea less chance for its work. It seems probable that when the harbor has filled in to a certain point, the equilibrium between the two acting forces will be again established, and it may be that this equality has already been reached, and no further filling will take place. The material washed into the harbor seems to have come from the beach adjoining on the south. The beach has grown in a northerly direction and crowded the channel many feet north of its former position. The westerly side of the harbor has suffered to the greatest extent, the sand having filled in to an average depth of eight feet over a large area. At one point, where formerly there was about four feet of water at low tide, the sand is now bare two hours after high water. The depth of water on the bar is also less than it formerly was. These changes which have taken place in the harbor have almost ruined it for the purposes for which it was formerly used.

The residents upon its shores derived some income from yachting parties which made it a resort in summer, and the fishing interest was not inconsiderable.

At present but few yachts come there, and the fishermen find it unsafe to venture far out, except in the best of weather, lest a storm coming up when the harbor is closed by low tide, they be caught without shelter.

This loss has caused the people interested in the harbor to feel very bitter against the "dykers," as they are called, and to hinder the success of the enterprise in every possible way. At one time they even tried to blow up the dyke. Three kegs of powder were obtained and placed in the sluice-way; they were exploded at dead of night, and though the dyke still remained intact, it was badly shaken. The perpetrators of this deed were not discovered, though parties who boast of having paid a share toward the powder are not wanting.

In conclusion I would say, that though the reclamation promises to be a success, it is, to a certain degree, at the expense of the harbor interest.

Though the pecuniary loss sustained by the damage to the harbor does not fall upon the owners of the marsh, it is, nevertheless, borne by the community at large, and is one of the

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drawbacks thoroughly to be considered before constructing a dyke of this kind.

This precaution is especially necessary, when, similarly to that at Marshfield, the marsh makes the main reservoir for tide-water, and is situated at the head of the estuary.

DRY CONCENTRATION AT FRISCO, UTAH.

By H. V. F. FURMAN, E.M.

THE Carbonate Mine is situated about two miles northwest of the town of Frisco, Utah. The vein is apparently a true fissure, having a dip of from 70° to 80°. The country rock is porphyry. The fissure, like those of the Bassick and Bull Domingo veins, has been filled in from the surface with rounded pebbles and boulders, which in this mine have been cemented together with galena, calc-spar, and in some cases with cerussite, which minerals have evidently been precipitated from a water-solution coming from below.[1] The walls generally have a casing of clay and talc. The shaft is sunk in the vein, and follows the foot-wall most of the way down. The vein has been cross-cut a distance of forty feet without any indications of a hanging-wall as yet. The ore-body is on the foot-wall, in some cases being very wide, so that it is worked by the system of square-set timbering. Owing to the structure of the vein, the ground is very treacherous, and requires careful and expensive timbering. It will be seen from the above brief description of the vein, that the ore as it comes from the mine contains a large percentage of waste, some of which can be separated by hand picking, but the larger portion of which can only be separated by crushing and subsequent mechanical treatment. The difficulty was the want of water, the water supply being derived from two wells which furnish just about sufficient to supply the boilers, the mine furnishing very little water. Hence, any wet process of concentration was out of the question, and the present concentrating works were erected. The main building is 93 feet 16 inches in length by 29 feet 6 inches in width. To this is attached the boiler and engine-house, a building 36 × 25 feet, and a two-story build-

[1] See also Professor Newberry's article on Ore Deposits in the QUARTERLY for March, 1880, pp. 101, 102.

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ing 19 × 15 feet, the ground floor of which is occupied by the Frue Vanner. There is also a building at the end, 26 × 12 feet, which is occupied by the crusher.

The ore is hoisted from the mine in buckets, and emptied into a car when it reaches the surface. The car is run around from the shaft-house over a trestle-work to the crushing-house, where the ore is dumped. Some of the ore is dumped in ore-houses near the shaft, where it is picked over by two men. The ore is fed into the crusher by a man who also looks after the dryer, rolls, screens, and elevators, and who receives $3.50 per day. From the crusher it passes directly into the drying cylinder, a form of dryer which is now used in many dry-crushing silver mills. The cylinder is iron, 24 feet 6 inches long, and having an inclination of 12 inches in that distance. On the inside of the cylinder there are two inch angle-irons, fastened the entire length of the cylinder, which raise the ore up as the cylinder revolves, and cause it to fall through the hot gases. A toothed wheel which encircles the centre of the cylinder works into a gear wheel, which is on a shaft driven by a pulley and belt. The cylinder is supported at each end by friction rollers. At the lower end of the cylinder is the furnace, which is built of masonry. The grate is 2 feet 4 inches wide by 4 feet 6 inches deep. The furnace burns two-thirds of a cord of dry pine slabs, or one and one-eighth cords of cedar wood in a twenty hours' run. The ore from the dryer falls on an iron riddle, 8 feet long and 21 inches wide, which is covered with a 10-mesh screen. The riddle is suspended at the upper end, has an inclination of 2 inches in 3 feet, and a throw of 4 inches in the direction of its length, which is given to it by an eccentric. The ore which passes through the screen falls into a chute, which takes it to the elevator. The ore passing over the screen goes through the first set of Cornish rolls. The rolls are 24 inches in diameter, have a face of 15 inches, and make twenty-eight revolutions per minute. The ore from the rolls falls on a riddle similar to the one before described. The ore which passes over the ten-mesh screen of the riddle passes through a second set of rolls similar to the first set. The crushed ore is then lifted to the screen room by means of an endless rubber belt, to which metal cups are attached, at a distance from each other of 9 inches.

The crushed ore is discharged upon a 20-mesh revolving screen. The ore passing through the screen falls into a chute

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which conveys it to a bin, from which it is hoisted to the 35-mesh screen. The ore passing over the screen at the lower end, falls into a chute which conveys it to the separators. In a similar manner the ore passing through this screen, and the succeeding ones, is elevated to the screen next in order, the ore passing through the last screen going to the Frue Vanner, whilst the ore which passes over the different screens is conveyed to the separators. There are six revolving screens in all, arranged in the following order: 20-mesh, 35-mesh, 50-mesh, 80-mesh, 110-mesh, and 140-mesh. The screens are all 8 feet 9 inches in length, and have an inclination of 1 inch in 1 foot. The 20-mesh and 35-mesh screens are iron wire, being cylindrical with a diameter of 32 inches. The other four screens are eight-sided, the sides being 1 foot wide and covered with silk screens. All of the screens revolve inside of a frame-work covered with cloth to retain the dust. The screen room is 35 feet 7 inches in length, by 12 feet in width. The bin room is directly underneath the screen room. The room in which the separators are located is on the second floor, underneath the bin room, and is 35 feet by 29 feet 6 inches. It contains eleven Paddock pneumatic separators.[1] Two separators run on stuff passing over the 20-mesh screen, three on that passing over the 35-mesh screen, two on that passing over the 50-mesh screen, one on that passing over the 80-mesh screen, two on that passing over the 110-mesh screen, and one on that passing over the 140-mesh screen. The machine consists of an inclined sieve (cloth), over which the ore passes, and through which air is blown into the ore by means of an elastic diaphragm below. The air current is intermittent, the number of puffs being regulated by cone pulleys, and the strength of the current being also proportioned according to the thickness of the ore-bed.

The ore is lifted by the air-current, and falls again on the sieve. By this treatment the ore, in passing through the machine, is gradually separated into layers on the sieve according to the specific gravity of the minerals which compose it. The crushed ore is delivered from above into a hopper, from which it passes to the sieve. As the ore flows down the sieve fresh supplies run in from the hopper, which is adjustable, so that the length of the bed and the time of treatment can be adjusted according to the ore treated. The inclination of the

[1] Described by John A. Church (S. of M., '67) in the Transactions of the American Institute of Mining Engineers, vol. viii., p. 148.

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bed, both lengthwise and sidewise, is controlled by means of screws at each of the four corners of the frame, and the air-chamber is made with flexible sides to permit this adjustment. The heads, middlings, and tailings are discharged from the end of the machine, over a tail-board, and run off side by side.

The horizontal movement necessary to place them side by side is effected by means of an inclined grating laid on the sieve. This grating consists of a series of narrow brass strips inclined to the path of the ore, which direct the concentrated ore to one side of the bed, where it collects in a trough and flows down to the tail-board. Upon this grating there is a similar one, made of broader strips, which are inclined in the opposite direction. The light mineral, being on top, is conveyed by this upper grating to a trough on the opposite side, and by the trough to the tail-board. The whole bed is inclined slightly toward the lower corner on the side of the heads. On the tail-board are two adjustable strips of wood, which prevent the heads, middlings, and tailings from mixing again as they are discharged into their respective boxes. The heads pass through a 1¼-inch gas-pipe to the packing barrels on the floor below, and the tailings in the same way to bins, from which they are drawn off into cars. The middlings are passed through the jigs again. The sieve consists of cloth, that on the jigs running on 10-mesh stuff being coarse, whilst that on the other machines is fine broadcloth. The coarse cloth has to be replaced every two to three months, whilst the fine cloth lasts from nine to ten months. There is a cloth hood over each of the machines, which is in communication with a No. 5 Sturtevant Blower, which draws the fine dust into a dust-chamber. The fine dust produced by the Cornish rolls is collected in a similar manner, and all the fine stuff is washed on the Frue Vanner. The pneumatic machine treating stuff between 110-mesh and 140-mesh makes only heads and tails, and the tails are washed on the Frue Vanner instead of being put through the machine again. There are three men to look after the machines: one to tend those treating 10-mesh stuff, one to tend those treating 20-mesh stuff, and one for the machines treating fine stuff. These men receive $3.25 per day each.

The Frue Vanner is situated on the ground floor. This treats all the fine stuff, but owing to the scarcity of water it can only be run part of the year, so there is a house in which the fine stuff which accumulates is stored. The water from the Frue Vanner has also to be saved, so the tailings are run off

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into settling tanks. There are five of these tanks, which communicate with each other, and from the last of which the water is pumped back to the Frue Vanner by a small hand-pump actuated by a connecting-rod and eccentric. On top of the water in the last tank there floats a scum of carbonate of lead, which is skimmed off from time to time. The man who tends the Frue Vanner also shovels the slimes out of the settling-tanks. This man receives $3.00 per day. The heads from the Frue Vanner are shovelled out of the settling-tank from time to time on to a drying-floor, and when partially dry are packed in half-barrels. The loss of water used on the Frue Vanner is about one hundred and fifty gallons per day. All the concentrates are packed in barrels, holding from 600 to 1,200 pounds, according to the purity of the concentrates, and shipped to the smelter, a distance of about two miles.

There are two engines, one of fifty-horse power, which runs the rolls, crusher, dryer, screens, elevators, and large blower (it is an ordinary horizontal, slide-valve engine, built by the "Erie City Iron Works"); the other, a five-horse power ordinary horizontal, slide-valve engine, which runs the separators, Frue Vanner, and No. 5 blower. There are two boilers, each 40 inches in diameter, with flues 16 inches in diameter and 20 feet long. Each boiler has two return flues, and is provided with a mud drum. The exhaust steam passes into the large water-tank, which has a capacity of 7,500 gallons. The boilers use about 8,000 gallons in a run of twenty hours, but a large quantity of this is saved by passing the exhaust into the water-tank. The boilers burn from four to five cords of cedar wood in a twenty hours' run. Cedar wood costs $2.50 per cord and pine slabs cost $3.25 per cord. There is one engineer to each shift, who receives $4.00 per day. There is also one man-of-all-work employed, who receives $3.00 per day, and one foreman. There are two shifts of ten hours each. The works treat about one ton of ore per hour. They have treated thirty tons a day; but the jig capacity is only about twenty tons, although the capacity of the crushing and screening machinery is much more. They use one gallon of oil a day, and two to three pounds of graphite grease a week on the machinery.

The ore as it comes from the mine will probably average 12 per cent. lead, 25 ounces silver, and 9 per cent. water. As the ore which comes from the mine is not carefully sampled it is impossible to say what percentage of the lead and silver is

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lost in the process of concentration. A calculation made on the best data shows in fact *no loss*, but over 100 per cent. gained.

The following table gives the results of the run for May, 1881:

640,820 *pounds of ore shipped from the mine to the concentrating works.*

Size mesh.	Product, pounds.	Assay value.	
		Lead, per cent.	Silver, ounces.
10	80,708	42.8	111.0
20	52,742	51.9	129.35
35	30,680	58.0	147.72
50	12,021	50.9	134.01
80	7,126	55.4	140.86
110	4,549	57.6	167.84
Dust, etc.	29,950	48.0	118.49

The concentrates from the Frue Vanner hold about 18 per cent. moisture.

The average of a large number of assays of tailings would give a trace of silver and no lead. The largest amount found in any of the samples of tailings was 1.45 ounces silver and a trace of lead.

The highest assay of tailings from the Frue Vanner gave 1.8 per cent. lead and 6.56 ounces silver, and the lowest assay gave 0.3 per cent. lead and 1.60 ounces silver. The time of running was 525 hours.

MILL RUN FOR JUNE, 1881.

621,835 *pounds of ore shipped from the mine to the concentrator.*

Size mesh.	Product, pounds.	Lead, per cent.	Silver, ounces.
10	80,979	30.8	68.97
20	34,835	45.6	105.86
35	14,835	44.6	104.91
50	9,574	30.1	88.21
80	6,302	43.0	97.70
110	3,842	57.2	134.30

The fine stuff treated on the Frue Vanner gave 22,838 pounds of concentrates, holding about 18 per cent. moisture, 43.3 per cent. lead, and 106.45 ounces silver to the ton.

The average of all the assays of the tailings from the Frue Vanner gave 0.2 per cent. lead and 1.45 ounces silver.

The average of all the assays made of the tailings from the Paddock separators showed a trace of silver and no lead by fire assay. Time of running, 520 hours.

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AURORA, NEVADA: A LITTLE OF ITS HISTORY, PAST AND PRESENT.

By the late HENRY G. CLARKE, E.M.

THE history of Aurora is that of the typical mining camp. The first discovery of ore was made in 1862, and in a few short months, Aurora, with a population of sixteen thousand people, was one of the liveliest mining towns Nevada has ever had; stamp-mills were erected, ore was plenty, and millions of dollars in bullion were produced; everybody had money, and saloons and dance-houses flourished without number.

For two years the "boom" lasted, till, in 1864, the rich ore that had been so plentiful in the croppings of the immense ledges that run through the district, seemed suddenly to be exhausted. The Comstock at this time was attracting a good deal of attention, and the population of Aurora literally emptied itself into Virginia City. Most of the mills were pulled down, and the lumber and machinery shipped away. Nearly all the wooden buildings shared the same fate, and nothing was left of poor Aurora but the brick buildings, which, for so young a town as Aurora then was, are remarkably fine ones, and the name of being the county-seat of Esmeralda County. A population of four or five hundred, who had confidence enough left to "stay with the proposition," as they say here, was all that remained.

In the cañon below the town may be seen the ruins of no less than seventeen mills, some of which were very fine ones, enclosed in brick buildings, every brick of which cost twenty-eight cents.

The machinery is still standing in the Del Monte Mill, which was one of the finest and most costly mills on the Pacific Coast. It affords a fine opportunity of comparison between the mills of twenty years ago and those of to-day. This mill was provided with twenty stamps, and was enclosed in a brick building that would have done credit to any Eastern city. Blake's rock-breakers were unknown when this mill was built, and the ore was broken and prepared for the batteries by being crushed

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under two immense stamps, weighing two thousand pounds each, and having a drop of 4 feet. There were no mortars, but a solid bed, or anvil, was surrounded by massive grates of bar iron, through which the quartz would drop as fast as broken. Each of these stamps was capable of crushing one ton of quartz per hour, but with an enormous expenditure of power, as is evident, when we consider that for each blow a ton weight of stamp was to be raised 4 feet. The stamps proper are seven hundred pounds weight, and all the wood-work about the batteries is nicely finished and oiled, a luxury not often seen now. The mill is provided with the first Wheeler pans ever cast, also the first Varney pans made were used in this mill.

In the two years of prosperity which Aurora enjoyed, she turned out over twelve millions in bullion. The geological formation in which the mines are situated is a belt of porphyry, which forms a succession of three or four hills, lying in a general northeast and southwest direction. This porphyry, disappearing under overlying formations both ways, but coming to the surface again ten miles southwest of here at Bodie, California, and forming the celebrated Bodie Bluff, on which the Standard, Bodie, Bulwer, and other well-known mines are situated. At Aurora this porphyry is seamed with numerous veins of quartz, whose croppings stand high above the surrounding country rock. These veins generally have a strike or trend of about N. 40° E., though there are some cross ledges running nearly east and west. The mineral occurs in the veins in bonanzas, or what are locally called rich chambers. The quartz for long distances will be barren, or nearly so, though the walls continue good, then suddenly it will become highly mineralized with native gold and silver and some sulphide of silver; at the same time the walls widen out, and the vein is filled from wall to wall with the richest kind of ore.

Most of the bonanzas that were worked in the old days outcropped at the surface, and this explains why the district produced so much bullion in the first two years of its existence. These chambers were literally "gouged" out, without any regard for the future of the mines.

After the rich chambers of ore had become exhausted and the money had been paid out in dividends and expensive lawsuits about titles, etc., work on the mines was abandoned without any attempt at prospecting the veins for new bonanzas.

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The sides of the hills below the outcroppings of these veins are covered with "float ore," and men make good wages to this day by breaking these quartz boulders and sorting out the ore.

Two or three years ago the Real Del Monte Company determined to prospect Last Chance Hill by sinking a deep vertical shaft and cross-cutting to the veins, none of which had ever been prospected below 200 feet from the surface. This work was inaugurated under the management of Mr. George Daly, but for the past year or so has been in charge of Mr. D. M. Tyrrell. In sinking this shaft large quantities of water were encountered, which, in spite of the large and powerful machinery put in, has finally caused the work to be suspended, and the problem of what these veins contain in depth is still an unsolved one. A project is now being considered by the company to run a tunnel from a point near the old Antelope Mill; this tunnel would be about four miles long, and would strike these veins at over 1,000 feet depth.

On Silver Hill, a New York company have purchased the Cortez Mine and commenced work under the management of the writer of this article, and have already uncovered a bonanza that bids fair to rival any of those worked in 1862–64. The Cortez and Utah claims are on a vein whose normal width, when not in bonanza, is about 3 feet. In the 1,500 feet of this ledge covered by the Utah patent, five bonanzas were found, which yielded in the aggregate over five millions.

In the Cortez ground, only 200 feet of the ledge has been prospected, and that only to a depth of 150 feet, this work opening up a chamber of ore, extending from the surface nearly to the 150 feet level, that was 50 to 75 long, the walls of the vein widening out to 20 and 25 feet apart. This chamber yielded two million dollars, and was worked out in 1862–64. The mine has been practically idle ever since, till the present owners came into possession, about April 1st of the present year.

Our first work was to commence sinking an incline in the vein from the bottom of the 150 feet level, it being the intention to sink to the 250 feet level and drift both ways. In sinking this incline, at a depth of 20 feet below the 150 feet level, the top of the bonanza was reached. The quartz which, down to this point had been very low grade, became highly mineralized

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with both gold and silver, and assaying from $250 to $1,000 per ton. The vein widened rapidly, and every foot added to the amount of gold that could be seen in the ore. At the present writing this incline has reached a depth of 50 feet, every foot having added both to the width of vein and grade of ore, pieces of quartz coming out that weigh 30 or 40 pounds, and assay $10,000 to the ton, the whole mass being literally filled with fine particles of gold. The proportion of the respective values in gold and silver in the Cortez ore is about two dollars in gold to one dollar in silver.

This discovery will probably cause some of the other veins that produced so much in the past to be further prospected, and so Aurora may regain her lost laurels.

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PROFESSIONAL NOTES.

Mining Engineering.

Cheap Shaft-sinking for Prospecting Purposes.—In the Pennsylvania hematite regions, the common method of prospecting is by means of small test-pits, sunk through the overlying clays and gravels to depths of from 10 to 50 feet, or more. The Pennsylvania Dutchmen who sink these pits often become quite expert at the business, and will take contracts at remarkably low rates to sink to any depth required, or until bed rock is reached. In the Lehigh Valley, according to Professor Cook, the contract price for digging such pits up to 50 feet in depth ranges from fifteen to twenty-five cents per foot. The peculiarity of the method followed by these prospectors is in the very small diameter of the shaft sunk, ranging from 30 inches to 3 feet only. A light pick and a spade, with handles about a foot long, are used for the work, and the dirt is hoisted in a small bucket, holding about fifty pounds, by means of a light windlass, which can easily be carried from place to place. Two good men can by this method sink 20 to 30 feet in a day, and the peculiar skill with which they adapt themselves to their narrow quarters is most remarkable. Up to a depth of 8 feet, or more, these human gophers sink without using the windlass, and contrive to throw the dirt clean out of the hole without so much as a pebble falling back on them. These pits, on account of their small diameter, and the care and skill with which they are sunk, apparently need no lining or timber-support to insure the safety of the men, whereas larger shafts, sunk in the ordinary way, would have to be timbered, and would cost five to ten times as much. A description of and notes on this method of prospecting will be found in the annual reports of the State Geologist of New Jersey for 1872 and subsequent years. Our attention was called to the method by Mr. William Allen Smith, '68, who has had occasion lately to employ two of these Pennsylvania shaft-sinkers in the examination of several hematite properties in this State.

Signalling in Shafts.—In the Osceola Copper Mine, Lake Superior, is a very ingenious system of signalling, well adapted for the system of hoisting in use. As in most metal mines, several levels are worked at once, and, consequently, the ore has to be hoisted from several different points in the shaft. The skip is loaded directly from the car, and to arrest and support it in the proper position for dumping the car-load from the level, a "gate," or heavy timber, is thrown across the shaft before the skip is lowered. The object of the system of signalling adopted is to inform the engineer at what level the skip will thus be stopped, as otherwise he must slow up as he lowers past each level, or run the risk of carrying away a gate and damaging the shaft-timbers and the skip.

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The signals are conveyed to the surface by a wooden rod, about two inches square, the different sections firmly joined together by iron straps, and supported by guides, etc., like the pump-rod of a Cornish pump. At the upper end, this rod is connected by an angle-bob with a horizontal rod leading into the engine-room, where it moves a long pointer over the face of a graduated dial. When this pointer is pushed back to zero it rings a bell. This is the signal to hoist. Before the skip reaches the surface, the pointer will probably be seen to move backward over the dial, stopping, we will say, at the figure indicating the fifth level. This has been done by the trammers at that level, who are waiting there with a car of ore, and inform the engineer that the skip should be lowered to that level next time. Some arrangement for locking the rod must, of course, be provided at each level, to insure that the rod takes the exact position to move the pointer to the proper figure on the dial in the engine-room. The rod being locked, also prevents the trammers at other levels from moving the index until the skip has been loaded at the fifth level. When this is done, the trammers unlock the rod, and by means of a lever push it upward. This moves the pointer in the engine-room back to zero and rings the bell, giving the signal to hoist. The rod now being free, can be moved by the trammers who may be waiting at another level.

Another form of Signalling Apparatus used in the shaft of the Koh-i-noor Gold Mining Co., Ballarat, is described in the report for 1880 of the Chief Inspector of Mines of Victoria.

This apparatus is used both as an indicator and a speaking-tube. A two-inch gas-pipe is placed down the shaft, which pipe revolves on a tapered step at the bottom, and is supported by glands resting on the top of plates fixed at intervals in the shaft. There are pins through the couplings to prevent the lengths of piping from unscrewing when moved. At each level, and at the surface, handles, with pointers attached, are clamped to the pipe, by which it can be turned on its axis. Alarm-bells are attached to the handles and indicating dials, either hoop-shaped semicircles, or flat horizontal discs, are provided at each level. The spaces on these dials are filled in with such words or messages as will be most useful. If more than one level is being worked at the same time, the indicator shows "Cage wanted at No. 1," and so on, as the case may be. When communicating from below, the handle is turned to the space marked "Attention," and, in moving, rings the bell. If the engineer be absent at the time the signal is given, he sees, on returning, by the pointer being at "Attention" that some one has been ringing. He then moves the handle, which rings the bell below. The man below then moves the pointer to the space on the dial indicating the article required. The handle is not moved again till something is wanted, when it is brought back to "Attention" as before.

For speaking purposes, a two-inch T coupling is fitted to the main pipe at each level, to which is fixed a rubber-tube with a tin mouth-piece. This tubing is of such a length that, when not in use, it hangs down, and thus prevents the sound from escaping.

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Experiments and Improvements in the Mines of Prussia during 1880.—An elaborate article, under a title similar to this, appears annually in the official journal. From this report [1] and from other sources we condense the following items of interest:

Mining Tools.—A jumper-drill with replaceable cutting-bits was introduced in 1879, and has since been used with success in the Königen Louise coal mine, in Upper Silesia.

[From the description given, this drill is apparently similar to the one invented by Capt. William Powell, one of the mining bosses at Drifton, and largely used by the miners in the Lehigh anthracite region since 1877. The chief advantage of Powell's drill is in the saving of labor in carrying the light bits, instead of the heavy drills, in and out of the mine to be sharpened. One of these drills may be seen in the School of Mines' collection of mining tools.]

Rotary Boring Machines, for cutting holes, or cross-cuts, through the pillars of coal, for ventilation, are now largely used in the Westphalian coal mines. These machines are worked by hand, two men at a time working a crank, motion being communicated to the cutter by gearing. Holes from twelve to twenty inches in diameter may be bored in this way. One of the more recent machines is arranged to cut an annular channel two and one-half inches wide, and thirteen and one-half inches in diameter. The core left by the machine is taken out by blasting with a small charge of powder in a central hole. Four men with this machine will bore twelve to fifteen yards of air-way in a shift.

Machine Drills.—The Frolich, Darlington, and other percussion-drills, have been used with good results, as in previous years.

Brandt's Hydraulic Boring Machine is a rotary machine for boring blast holes, with an annular steel cutting-bit, leaving a core like the diamond drill. The rods and bit are revolved by two small hydraulic engines, at a speed of about thirty turns per minute, while at the same time the bit is forced against the rock by strong hydraulic pressure. The waste water from the hydraulic engines is forced through the hollow drill-rods, and serves to keep the bit cool, and, at the same time, wash the *débris* of the boring from the hole. This machine has been introduced in several coal mines in Westphalia and Rhenish Prussia. The first experiments were not successful, owing to the defective shape of the steel cutting bit, which did not give sufficient clearance for the rods to work. This remedied, further experiments at several mines showed more satisfactory results. In hard sandstone conglomerate the drifting by machine costs but about sixty per cent. as much as by hand labor. In soft shale the cost was nearly equal, with sometimes a slight difference in favor of hand-drilling. The progress of the drifts per month—fifty-five to seventy-five yards, according to character of rock—was considered satisfactory.

[1] Zeit. für d. Berg-, Hütten-, u. Salinen-Wesen, B. 29, pp. 238–276, 1881. See also Berg- u. Hütten-männische Zeitung, Sept. 30, 1881, p. 373.

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This machine is, of course, especially applicable when hydraulic power is available, and has the advantage of not requiring a very elaborate or expensive plant, a pipe or column only being necessary, leading from some convenient reservoir at the surface to the working place below. One-fourth cubic foot of water per minute only is needed to run the machine.

Compressed Blasting Powder, in the shape of hollow cylinders two and one-half inches long, and of different diameters, according to size of drill-holes. The requisite number of these cylinders are placed in a cartridge-shell, the fuse is inserted in the small central channel, and the charge tamped and fired in the usual way. Sometimes a small detonating cartridge containing dynamite, fulminating mercury, or a picrate salt, is attached to the fuse, and inserted in the central channel, with good results. The results with this powder are varied. At two coal mines in Upper Silesia the experiments were satisfactory, the blasts giving a greater proportion of large coal and less culm than with ordinary powder, with less smoke and greater safety in handling.

Wood Powder.—A new powder, apparently a nitro-cellulose compound, has been tried with rather unsatisfactory results.

Gun-Cotton.—Experiments more successful than formerly, but it has been definitely proved that it must be kept dry to give good results.

Blasting-Gelatine and Gelatine-Dynamite.—These new explosives, introduced by Nobel, the inventor and manufacture of nitro-glycerine compounds, have, during the last two or three years, attracted much attention. Blasting-gelatine consists of nitro-glycerine converted into a semi-solid by dissolved gun-cotton, and rendered less sensitive to shocks by a small admixture of camphor. Gelatine-dynamite is a mixture of this explosive with nitrate of potash. From a series of practical tests at the Friedrich's Mine, at Tarnowitz, in hard and compact rock, the blasting-gelatine has been proved to be safer, more effective, and, notwithstanding its higher price (fifty per cent. more), more economical than ordinary No. 1 dynamite.

The gelatine-dynamite is sold at the same price as ordinary No. 1 dynamite, but is, it seems, not likely to replace the latter. It requires a strong detonating cap, containing not less than eight grains of fulminate of mercury, otherwise the full force of the explosion is not developed, and even then its effect is not greater than that of ordinary dynamite. In case the initial detonation is insufficient, the smoke of the explosion contains so much undecomposed nitric acid gas as to be irrespirable.

A series of practical tests, extending over several months, of the value of blasting-gelatine as compared with ordinary No. 1 dynamite, were made at Mansfeld. In drifting in conglomerate, about one-third per cent. more rock was excavated per day's labor, and the cost for explosive for equal volume of rock was about one-third less when the blasting-gelatine was used. In softer rock the difference was not so great, but still showed economy in the use of the stronger explosive, notwithstanding its cost. Nine holes, charged with the new explosive,

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did the work of twelve charged with dynamite. Blasting-gelatine has also the advantage of not freezing as easily as ordinary dynamite, and of being unaffected by water. Explosions at the Rhein-Preussen Mine, and at Leimbach, near Mansfeld, have thrown some doubt on the safety of this material, as these explosions are said to have been spontaneous. This, however, has not been clearly established.

Blasting-Pods.—A new method of firing blasts has been introduced at the Gräfin-Laura coal mine in Upper Silesia. A paper or cardboard tube, about an eighth of an inch in diameter, and perforated with small holes, is placed in the cartridge so as to leave an open channel in the middle of the powder, and thus secure a more perfect ignition than when fired with a fuse or straw. An economy of twenty-five per cent. of powder, and a larger percentage of large coal, is said to result from the use of this simple device. [The Gomez fuse, which was used to some extent in the Pennsylvania anthracite mines several years ago, had a similar object. This consisted of a flat tape of folded cartridge-paper, charged with some quick explosive mixture, possibly one of the picrate salts. On lighting one end of this fuse it exploded at once through its whole length, whether the piece was 3 inches or 3 feet in length, with a report like a fire-cracker. In using, it was intended that the fuse should be inserted in the powder, reaching to the bottom of the cartridge, the object being to explode the whole charge simultaneously, and thus avoid loss of energy by the escape of part of the gases through cracks formed before the whole charge is exploded. Some experiments made by the writer seemed to prove that there is but little, if any, advantage gained by the use of the fuse. In a few cases there was an apparent gain of twenty to twenty-five per cent., the fuse doing the work with about one-fifth less powder than was judged necessary. With another miner, who perhaps estimated the amount of powder required more accurately, a similar reduction in amount caused the shot to fail entirely. It would thus seem that the gain, if any, lies in the saving of the extra amount of powder put in by the miners "to make sure," and it is probable that the same economy might be obtained with any fuse or peculiar method of tamping or firing, provided that the miners were instructed at the same time to lessen by one-fifth the amount of powder used.]

Underground Haulage.—Iron cross-ties and longitudinal iron sleepers have been used for surface and underground tracks with good results, costing but little more than wood, being more durable, and demanding less work to keep the track in good condition.

Electric Railway.—In the Senftenberg mine, in the Brandenburg district, an attempt was made to introduce a dynamo-electric locomotive for underground haulage. On account of the low temperature of the air in the gangways, it was found impossible to keep the middle iron rails free enough from snow and ice to maintain electric connections, and the experiment was therefore unsuccessful. When the rail was in good condition a useful effect of forty-seven per cent. was obtained.

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Self-acting Incline with Endless Chain.—In the Ziewald section of the government coal mine König at Saarbrücken, a plane has been constructed which is worked by the endless-chain system. This system was adopted on account of the length of the incline, about one-third of a mile long, and the slight dip of the coal seam, four degrees only (or about seven feet in one hundred). At the upper end of the plane the chain passes around a horizontal brake-wheel, with screw-and-lever brake of English design. At the lower end is a tightening-wheel to keep the chain taut. The chain weighs about five and a half pounds per yard, and has a breaking strength of three and three-fourths tons. The links are about two and one-half inches long and of half-inch iron. The cars are run under the chains, one at a time, at intervals of fifteen metres, and fastened to the chain by forks, attached to the upper edge of the car-body. In order to secure the regular placing of the cars, a bent lever is inserted in a slot cut in one rail at fifteen metres from the landing. One end of this lever projects slightly above the surface of the track, being held in that position by a spring. The other end is attached to a bell-wire. The car passing over this arrangement rings the bell and gives the lander the signal to attach another car to the chain. The chain is thus supported on a line of cars, the loaded cars on one track, and empty cars, attached in like manner at the bottom of the plane, on the other. The speed of the chain is regulated by the brake at the top. The plane has been in operation since July, 1880, and at the time of writing, the mine not having been fully developed, 500 wagons, or 250 tons of coal, are lowered per shift at a speed of about two feet per second. When by the further development of the mine, the full capacity of the plane is demanded, it is proposed that the velocity shall be one metre per second, or four wagons per minute. The capacity of the plane will then be 850 tons per eight-hour shift.

The force of men required is six "hitchers-on" and one brakeman. This force is necessary on account of there being a station at the middle of the plane.

(To be continued.)

Colorado Minerals.

As but little has been published heretofore respecting the mineralogy of Colorado, Dr. Malvern W. Iles, '75, sends us the following list of some of the rarer Colorado minerals which have come under his notice. Dr. Iles refers to the "List of Colorado Minerals," by Dr. G. Alden Smith, as being the most complete yet published; many of the following species, however, are omitted, and indeed, in many cases, have not heretofore been known to occur in the State.

Tellurium.—Native tellurium in dendritic form, *altaite*, *hessite*, *sylvanite*, and *calaverite* have been found in Boulder County.

Sulphur.—Handsome samples of native sulphur are obtained from the celebrated Iron Mine, Leadville.

Alumina.—As a general thing, the ores about Leadville do not contain a large amount of alumina. According to Dr. Hillebrand, chemist to the State Geo-

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logical Survey, the Adelaide gouge matter contains a large percentage of sulphate of alnmina.

Specimens of *mendozite* have recently been found by Dr. Iles, at Red Cliff, Col. The mineral occurs as a warty excrescence upon a siliceous gangue, readily soluble in water, the solution having the characteristic alum taste. The salt was treated with water, boiled, filtered, and evaporated to crystallization, and then analyzed, yielding the following result:

	Found.		Theoret.	
SO_3	38.35	per cent.	36.30	per cent.
Al_2O_3	14.66	"	11.70	"
NaO	4.35	"	7.10	"
H_2O (Dif.)	[42.64]	"	44.90	"
	100.00		100.00	

This analysis establishes the formula $NaO.SO_3+Al_2O_3.3SO_3+22H_2O$ (see Dana's "Mineralogy," p. 653.)

Pyromorphite.—This mineral has been found in the Sliver Mine, Leadville, in handsome specimens.

Tin.—Dr. Hillebrand has found traces of tin in a number of the Leadville ores; the bullion also shows some tin.

Arsenic.—It is safe to say that most of the ores in this immediate district contain some arsenic. Very good specimens of *realgar* have been found in the Evening Star Mine.

Molybdenite.—Specimens have been found on Upper California Gulch, near Leadville, and at Pitkin.

Bismuthinite.—Specimens have been found at Gold Hill, Boulder Co.; also in Gilpin, Park, and Lake Counties. A mill-run sample of one hundred pounds, from ore near Leadville, yielded seven per cent. bismuth.

Uraninite.—Professor Pierce, of the Boston Colorado Works, has discovered a well-defined vein of this mineral above Central City.

Manganese minerals.—A number of the mines at Leadville contain a large percentage of manganese, occurring as oxides. The amount has been found to vary from a trace to as high as forty-nine per cent. of the metal. Several of the mines have shipped lots, varying from fifty to one hundred tons, which contain a greater percentage of manganese than iron. A large percentage has been noticed in certain lots from the Little Chief, Little Pittsburgh, Glass Pendry, Silver Wave, Long and Derry Mine, the Henriett, and some others.

Immediately above the city of Leadville, on the California Gulch, are immense quantities of manganese ore, valueless so far as lead and silver are concerned, but which Dr. Iles thinks could be utilized in certain industries. As an illustration of a manganiferous ore, he cites the following analysis of a mill-run sample from the Long and Derry property:

SiO_2	25.30	per cent.
Fe_2O_3	19.85	"
Al_2O_3	1.75	"
Mn_3O_4	39.10	"
$CaCO_3$	3.57	"
PbS	10.34	"
	99.91	"

Dr. Iles recently discovered a new manganese mineral, which has been partially described in the *Mining Index* of Denver, Col., from which we extract the following:

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"The locality in which it is found is Hall Valley, Park County, Colorado. The specimens in question were from the McDonnell property, Middle Swan Creek, thirteen miles from Webster Station. The vein containing the same outcrops above timber-line, and the apex is covered with frozen snow. The gangue is said to be about three hundred feet wide, and the surface rocks adjacent are encrusted with the substance. There is a pure streak of this mineral, from two to eight inches in width. The developments consist of a tunnel, forty feet long, and a shaft, twenty-five feet in depth. In both of these openings the presence of the white soluble substance is everywhere apparent.

"An analysis of the mineral by Dr. Iles shows the following composition:

Protoxide of iron	4.55	per cent.
Oxide of zinc	5.63	"
Sulphuric acid	35.85	"
Protoxide of manganese	23.18	"
Water	30.18	"
Total	99.39	per cent.

Formula: $(FeO.ZnO)\ SO_3 + 2\ MnO.SO_3 + 12\ H_2O$.

"It is a white, friable mineral, with a bitter, astringent taste, readily soluble, even in cold water; crystallizes in thickened prisms, belonging to the orthorhombic system. It is loosely adherent to a siliceous gangue, which was found to contain iron sulphides and zinc sulphides; also some silicate of zinc, and is thought to be a product resulting from an oxidation of the above sulphides."

The *Mining Index* suggests that this mineral receive the name of "*Ilesite*," which name THE QUARTERLY hopes will be accepted by mineralogists.

Copper.—At Aspen, Roaring Forks District, copper occurs as azurite and malachite, associated with barite, which is very similar to the occurrence of these minerals at Cheshire, Conn. The Miner Boy contains from three to seven per cent. metallic copper, in the form of azurite, malachite, and chalcopyrite. The Little Johnny contains a much larger amount of copper in the form of carbonates. The ore from the Farwell Consolidation, in Independence Gulch, forty miles from Leadville, contains chalcopyrite; the concentrates contain about three per cent. metallic copper. The iron-pyrites district, at Ward, Boulder County, contains handsome specimens of dendritic copper. The Robinson ore contains chalcopyrite. The Dyer Lode contains the carbonates of copper.

Pyrargyrite.—Antimonial silver occurs in considerable quantities in the Columbia Mine and in the Big Pittsburgh.

Of the commoner minerals, Dr. Iles mentions the following:

Epidote.—This species occurs, associated with fluorite, in several localities in Gunnison County.

Barite.—This mineral is found in the R. E. Lee, Hibernia, Evening Star, Catalpa, Chrysolite, and other Leadville mines. Most of the Aspen and Montezuma ores also contain heavy spar.

Fluorite.—This species is found at Hall Valley and in numerous other parts of the State.

Apatite.—This mineral is remarkable on account of its occurring in twin tabular crystals.

Orthoclase—This feldspar, generally in twin crystals, occurs in very great quantities on Gregory Hill, Black Hawk.

Olivine.—This mineral has been found in moderately large quantities near Poncha Springs.

Cerussite, *anglesite*, *lannarkite*, *cerargyrite*, *limonite*, *hematite*, *bog iron*, *calcite*, *dolomite*, and other of the commoner minerals are largely found throughout the State.

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In addition to the above, Dr. Iles briefly mentions the following artificial minerals which he has noticed in connection with lead smelting:

Chlorobromide of Lead.—See QUARTERLY, vol. ii., No. 4, p. 211, and *American Chemical Journal*, vol. iii., No. 1, March, 1881.

Artificial Anglesite.—This has been detected six feet above the slag tap-hole, associated with chlorobromide of lead. Analysis showed the product to be pure sulphate of lead, with the exception of a small amount of iron and silica.

Manganese.—A beautiful rose-colored sublimate, found to be chiefly manganese and sulphur, has been noticed, at very rare intervals, forming on the exterior of the sectional water-jackets near the joints.

Arsenic.—Globules of metallic arsenic have been noticed in the slag.

Sulphur, realgar, litharge, minium, hopper-shaped crystals of *galena*, and finally *subsulphide of lead*, have been found in the furnace-hangings.

MECHANICAL ENGINEERING.

Tests of Non-conducting Materials.—Mr. Charles E. Emery, of New York, has recently made some experiments upon relative non-conductivity, with reference to the needs of the companies who propose to furnish steam for heat and power through underground pipes. His apparatus consisted of a boiler, 12 feet long and 4 feet in diameter, with three 10-inch flues passing through it. Inside these flues were smaller tubes, through which the steam passed. The non-conductors surrounded the inner tubes, and water was kept circulating around the flues in the outer shell. The results were shown by the amount of steam condensed in the inner tubes, care being taken to drain the steam-pipes before they reached the apparatus, and to supply dry steam from a draining-drum just below the apparatus. Radiation was also reduced to a minimum. While the series contains some elaborate detail, which cannot be discussed here, the following results may be of interest. A layer of hair-felt, two inches thick, gave the best result, and, using equal thicknesses of the other materials, the following table gives the results in percentages:

Material.	Non-conductivity.
Hair-felt	100.
Mineral wool, No. 2	83.2
" " " and tar	71.5
Sawdust	68.
Mineral wool, No. 1	67.6
Charcoal	63.2
Pine wood, across grain	55.3
Loam	55.
Gas-works lime, slaked	48.
Asbestos	36.3
Coal ashes	34.5
Fuel coke	27.7
Air-space, 2 inches deep	13.6

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The mixture of tar with the mineral wool is made to water-proof it. The tar is painted on the pipe, and volatilizes under heat and penetrates the wool. It will then no longer absorb moisture. The low result from air-space no doubt is due to the unimpeded circulation of the currents. Where they are stopped, much better results will follow.

On the Use of Mineral Wool.—The writer has had occasion recently to inspect several long runs of pipe which were protected from radiation by mineral wool. The wool was tightly packed around the pipes, and was encased in a sheathing of galvanized iron. This sheathing was made in rings, and was put together like a stove-pipe, but with the ring-seams soldered. The hangers by which the pipes were suspended passed through the top of the sheathing. Parts of the runs were within doors, but a part in the open air was protected from the weather only by a platform over the sills from which the pipes were hung. It was possible, therefore, for rain and snow to beat in upon the sheathing, and to find their way through the hanger-vents to the mineral wool within. Melting snow was the worst enemy, for its attacks were more persistent. The sheathing was just warm to the touch, and rain dried up rapidly. The pressure in the pipes was about sixty pounds. But the greatest difficulty was caused by the tendency of moisture within the sheathing to travel a considerable distance from where it entered. The presence of moisture, with finely divided slag containing lime, favors corrosion and external pitting of the pipes; the water collects in pockets in the sheathing, and, if vaporized by the heat, finds difficulty in reaching a vent. In one of these pockets the water could be squeezed copiously from the wool, from which the heat had not been able to expel it on account of the situation of the pocket. The pipe at that point will have to be renewed, so great is the corrosion. It was the suggestion of the writer to cut the sheathing into sections of twelve feet or so, and to unite the sections at the breaks by "bandages" of galvanized iron, formed into a ring by light bolts through flanges at the longitudinal joint. This longitudinal seam should be slightly open, so that any travelling moisture could find a ready vent, and not pass from section to section. The continuous sheath is obviously unwise, and varying qualities of mineral wool may make it dangerous. Acidity or alkalinity of the wool will cause corrosion when aided by heat and moisture.

The writer has also had occasion to criticise a non-conducting cement made of asbestos and hair, mixed to a paste with plaster and mortar. Although put on in a layer one and one-half inch thick, the exterior of the lagging is too warm to be economical. It cracks unpleasantly sometimes, but does not seem to attack the metal when kept dry.

A very good non-conductor was put forward last year. It is the furze from the head of the swamp "cat-tail." With a conductivity about as low as hair-felt, it is without the drawbacks of the latter of odor and decay. It is too recent to give quantitative results upon it as yet.

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SCHOOL OF MINES NOTES.

FOR the last few weeks the School of Mines post-office—or rather the shelves and bulletin-boards which serve that purpose—have been loaded with packages bearing the official stamp of the Treasury Department, and addressed to students of the higher classes in the School, these packages containing copies of the annual report of the Director of the Mint. This exceedingly valuable publication is sent free of charge to all who make application, by letter or otherwise, to Horatio C. Burchard, Director of the Mint, Washington, D. C.

MR. WILLIAM PISTOR, E.M. (S. of M., '68), who has been so long connected with the School as Instructor in Drawing, and to whom we are indebted for much of the efficiency of the present admirable system of instruction in free-hand and mechanical drawing, has resigned his position in the School, in order to be able to devote his whole time to his profession. Mr. Pistor is now a member of the firm of Hamilton & Pistor, Architects, at No. 35 Broad Street, of this city.

MR. A. D. CHURCHILL, E.M. (S. of M., '80), has received the appointment of Instructor in Drawing, in the place of Mr. Pistor, having for some time past held the position of Assistant. Mr. J. L. Greenleaf, C.E., of the same class, is now Assistant in Drawing, taking the position left vacant by the promotion of Mr. Churchill. We are confident that under the direction of Messrs. Churchill and Greenleaf, the work of the department will be as systematic and the instruction as thorough as heretofore.

COLUMBIA COLLEGE has recently had a most important addition to its endowment in the shape of a bequest from the late Stephen Whitney Phœnix, of a sum variously reported at $500,000, $600,000, and upward. We understand however, that one or two members of the family of Mr. Phœnix have the use of the income from this money during their lifetime, so that the college does not at once profit by the bequest. Mr. Phœnix, we are informed, attached no conditions to his gift, but expressed a wish that a portion of the money should be expended in the fitting up of a new and well-equipped laboratory for the Department of Physics.

The total valuation of the property belonging to Columbia College, including the old and present sites and the Botanic Garden property, amounts, according to a report just finished, to $6,089,000, from which an income of $332,133 is derived. It is not generally known or appreciated, however, how recent has been the prosperity of the college. Up to 1865, the valuable up-town property, the "Botanic Garden," presented by the State in 1814, was a constant source of expense, it having been estimated that it has cost in taxes, assessments, interest, and other items, a round half-million of dollars.

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For many years the revenues of the college were absurdly inadequate for its needs. From the years 1850–55, the revenue of the college applicable to paying salaries and running expenses was not more than $25,000, and one year it fell below $21,000. 1866 was the first year in which a deficit had not to be made up by mortgaging the college's property or appealing to the generosity of the college's rich patrons. A debt was thus accumulated, for the interest upon which the college paid in 1862 more than $13,000. In 1862, the income was about $70,000, chiefly derived from the down-town property, and the deficit for the year was $4,324. Since 1866, the work of the college has widened in proportion with its income, so that there has not been so large a surplus, nor has the college been so enormously rich as is sometimes supposed.

THE history of the Botanic Garden tract, to which Columbia owes its present wealth, is sufficiently curious. In 1768, a medical school had been founded in connection with the college, and about the beginning of the century the State of New York, being appealed to for aid on behalf of the Medical School, decided to buy the Botanic Garden of a certain Mr. Hoosac, whose hobby was botany, and present it to the Medical School, upon condition that the school would keep the garden in repair and distribute botanical specimens. The Medical School took the gift, but found before long that it had a white elephant on its hands, which cost money and returned nothing, and as the prospect of being able to sell or rent the land was not encouraging, the school asked the State to take it back again, and was glad to have the offer accepted. The State had the Botanic Garden on its hands for a number of years, and when Columbia College asked for something in exchange for the 24,000 acres granted by the State of New York in 1767, but afterward found to be in Vermont, the State "jumped" at the chance of giving to the college its Botanic Garden, and the college accepted. This was in 1814.

For the next forty years the Botanic Garden was a constant source of expense to Columbia College. When the city's growth reached the Botanic Garden, and streets were cut through it, it was found that the property included the blocks bounded by Forty-seventh and Fifty-first Streets and Fifth and Sixth Avenues. The Sixth Avenue front, however, does not belong to the college. In 1863, the trustees received an offer from the Collegiate Dutch Reformed Church of $80,000 for sixteen lots, including eight lots facing Fifth Avenue, between Forty-eighth and Forty-ninth Streets, and four lots on each of those streets. The offer was accepted, and when President Barnard was elected President, in 1864, the trustees had under discussion a request on the part of the church trustees to cancel the contract. The college trustees refused to annul it, and the result was that, before the church trustees had erected a building on half of their plot, they had been enabled to sell the remaining half for $134,000, thus getting their ground for nothing and $54,000 besides. In 1860, there was only one house on the college plot—that built by the late Dr. Ward, who was much criticised for erecting so handsome a house "miles out of town." It is related that one day, when Dr. Ward was looking at the building going up, some one asked him, "What fool was putting up a palace in the wilderness?" In 1865, the site of the old

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Botanic Garden, which had been so long a burden to the college, began to be much sought after, and was judiciously leased. The property down-town, which was leased in 1857, when the college moved to its present site, also became extremely valuable, and a surplus grew rapidly, which soon wiped out the debt which had been accumulating for a number of years.—*From a report of an interview with President Barnard: Evening Post, December* 7, 1881.

THE SUMMER CLASS IN MECHANICAL ENGINEERING OF 1881.—Under the very able direction of Mr. F. R. Hutton, the summer class in mechanical engineering held its fourth annual session, extending over a period of four weeks from May 30 to June 25, 1881. Before relating the experience of the class, a few words about the objects of the course will not be amiss. These are three in number. First, to teach the student that knowledge of the processes of the machine-shop without which he cannot intelligently design pieces of machinery or other iron-work which, in his practice as an engineer, he may be called upon to construct. If ignorant of these processes, he will be apt to make designs costing more than others answering the same purposes designed by his rival, who possesses this information. To have this knowledge of when and how economy of construction can be secured, is, therefore, an important factor in the success of a young engineer. It will also enable him to estimate the approximate cost of work, and to judge of the advisability of accepting or proposing certain contracts. Secondly, to give him a general knowledge of the construction and management of a machine-shop, so that he himself shall know how to build and superintend one. To construct a shop he must be acquainted with the styles of buildings employed, their foundations, and the thicknesses of walls adapted to different purposes; he must know the kind and number of machines necessary to do certain classes of work, the amount of space required by each machine, and also the most advantageous arrangement. That he may superintend a shop, he must be able to distinguish between good and bad work, and he must know how much time is required and how many men are necessary for doing a given amount of work.

Thirdly, and not less important than either of the other objects, the class is designed to teach the student a systematic plan of investigation. By this means, when entering a machine-shop, he will know what to look for and where to look for it. Having gained a knowledge of what is common practice, he will be able to direct his attention to those devices which are new to him, and which may have some advantage over practice elsewhere. The course also forms a preliminary to the discussion of theoretical kinematics, during the third year, those who attend it knowing at once the practical application of many of the principles deduced.

These are briefly the objects of the summer school, and to attain these, the pioneer class of volunteers, consisting of thirteen men, under Mr. Hutton, entered the shops in June, 1878. From the start the class was successful; it was immediately legalized by the trustees, and similar classes have been formed each year. To say that all these qualities are attained by the student in four weeks would be vain, and would cast a slur upon the engineer's profession. Still, the amount of information which can be obtained by faithful workers, when properly

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directed, is almost incredible. At the close of the summer vacation an illustrated memoir is required from each student, and to examine these will convince any one of the great benefit derived from the school. One of the main features of the school, and one which certainly has done a great deal toward bringing about its success, is the fact that it is entirely optional. Those most interested in the study of engineering will always take the course, and the summer class will thus be made up of earnest workers, whose work will not be obstructed or delayed by those who have little or no wish to learn. Of course we do not mean that all those who do not take the course are not studious, for it often happens that men who are anxious to join are prevented from doing so by circumstances. On the contrary, if the course were obligatory, some would attend who, by the disturbance and annoyance they would create, would soon put an end to the aid and co-operation of the proprietors of shops, without which the school cannot exist. We hope, therefore, that this course will be continued as optional, and that other practical courses will be added, as this combination of the study of theory and practice cannot but increase the reputation of the school and the usefulness of its graduates.

To come back to last summer's class. The works visited were, as in previous years, the Quintard, the Morgan, and the Delamater Iron Works, and to the proprietors, superintendents, and workmen, the class is greatly indebted for their courtesy and assistance. There were in all twenty students in attendance, representing the "student element" of '83. They were divided into two squads, of ten each, and these again into pairs, working independently of each other. During the first two weeks one squad was at Quintard's and the other at Morgan's; during the third they exchanged quarters. While at these works, Mr. Hutton would spend the forenoon with one squad and the afternoon with the other, delivering to each a lecture in some convenient nook. These lectures embraced a general discussion of the whole subject, and also directed the attention of the students to what was good and what was bad practice, and to that which it was particularly important to observe. The students, provided with printed copies of an exhaustive and systematic outline of study prepared by Mr. Hutton, then scattered, each pair taking sketches and notes of whatever they deemed important. At noon an hour was devoted to lunch and, at the same time, to the asking of questions and to discussions on what had been seen. The afternoon was spent in the same manner as the morning. During the fourth week both squads were at Delamater's, and on the 25th of June the class separated, everybody seeming to be thoroughly satisfied with the manner in which he had spent the first four weeks of his vacation. The memoirs handed in this fall were quite voluminous as compared with those of previous years. The drawings especially were in many cases executed with a neatness and regard for accuracy of detail which showed, in a striking manner, the value of the thorough training in free-hand and mechanical drawing during the first and second years of the school course.

In conclusion, we would suggest that, in addition to the memoirs required upon the whole subject, each student should be requested to investigate and write a short memoir upon some special subject assigned to him, as, for example, "Shafting," including pulleys, gear-wheels, belts, etc.; "Foundations for Cranes and Steam Hammers,"

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"Boilers," "Blowers," etc. These could be read before meetings of the class, or before the Engineering Society. They could not fail to be very instructive, and would amply repay students for the time spent upon them. Should this plan be carried out, THE QUARTERLY will print any of such papers which may be worthy of preservation in permanent form.

BOOKS ADDED TO THE SCHOOL OF MINES LIBRARY SINCE OCTOBER, 1881.

MINING AND MECHANICAL ENGINEERING.

Burat, A. Épuration de la Houille. 4to, 22 pp. Paris, 1881.

Berg- und Hüttenmännisches Jahrbuch der K. K. Bergakademien zu Leoben und Pribram und der Königlich ungarischen Bergakademie zu Schemnitz. 8vo. Wien.

Edwards, Emory. Modern American Marine Engines, Boilers, and Screw Propellers. 4to, 146 pp. 30 Plates.

Michigan. Annual Review of the Iron Mining and other Industries of the Upper Peninsula of Michigan. A. P. Swineford. 12mo, 177 pp. 1881.

——— Annual Report of the Commissioner of Mineral Statistics of the State of Michigan for 1879. 8vo, 28 pp. Map. 1881.

Pennsylvania. Reports of the Inspector of Mines of Anthracite Coal Regions, 1870, 1871, 1872, 1877.

GEOLOGY AND NATURAL HISTORY.

Annals and Magazine of Natural History. Vol. 7. London, 1881.

Bert, Paul. Premières notions de Zoologie. 12mo. 229 pp. Paris, 1881.

Brogniart, Adolph. Recherche sur les Graines Fossiles Silicifiées. 4to, 93 pp. Paris, 1881.

Caswell, J. H. Microscopic Petrography of the Black Hills of Dakota. 4to, 56 pp. Washington, 1876.

Emerton, James H. Life on the Sea-shore. 8vo, 143 pp. Salem, 1880.

Habirshaw, Frederick. Catalogue of the Diatomaceæ.

Hervey, A. B. Sea Mosses. 8vo, 281 pp. Boston, 1881.

Meunier, Stanislaus. Première notions de Géologie. 12mo, 158 pp. Paris, 1881.

Packard, A. S., Jr. Zoölogy. 8vo, 719 pp. New York, 1880.

——— Guide to the Study of Insects. 8vo, 715 pp. New York, 1880.

Reinsch, Paulus Friedrich. Neue untersuchungen über die Mikrostruktur der Steinkohle des Carbon, der Dyas und Trias. 4to, 124 pp. Leipzig, 1881.

Renault, M. B. Cours de Botanique Fossile. 8vo, 185 pp. Paris, 1881.

Sachsse, Dr. Robert. Phytochemische Untersuchungen. 8vo, 119 pp. Leipzig, 1880.

Semper, Karl. The Natural Conditions of Existence as they Affect Animal Life. 8vo, 472 pp. London, 1881.

Shaler, N. S., and W. M. Davis. Glaciers. 4to, 196 pp. Boston, 1881.

U. S. Geological Survey. First Annual Report. By Clarence King, Director. 8vo, 79 pp. Washington, 1880.

——— Geology of The Black Hills of Dakota. 4to, 566 pp. and Atlas. Washington, 1880.

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U. S. Geographical and Geological Survey of the Territories: Idaho and Wyoming. By F. V. Hayden. Eleventh Annual Report. 8vo, 720 pp. Washington, 1879.

U. S. Geological and Geographical Survey of the Rocky Mountains Region. Geology of the High Plateaus of Utah. 4to, 307 pp. Washington, 1880.

CHEMISTRY AND PHYSICS.

Arendt, Dr. Rudolph. Technik der Experimental Chemie. 8vo. Leipzig.

Armstrong, Henry E. Introduction to the Study of Organic Chemistry. 12mo, 349 pp. New York, 1880.

Cristiani, R. S. A Technical Treatise on Soap and Candles. 8vo, 581 pp. Philadelphia, 1881.

Sawyer, William Edward. Electric Lighting by Incandescence, and its Application to Interior Illumination. 8vo, 185 pp. New York, 1881.

Tucker, John H. A Manual of Sugar Analysis. 8vo, 353 pp. New York, 1881.

METALLURGY.

Jeans, J. S. Steel: its History, Manufacture, Properties, and Uses. 8vo, 860 pp. London, 1880.

Randol, J. B. Production and Price of Quicksilver for Thirty-one Years. San Francisco, 1881.

Reiser, Fridolin. Das Härten des Stahles in Theorie und Praxis. 8vo, 136 pp. Leipzig, 1881.

Stegman, H. Gasfeuerung und Gasöfen. 8vo, 266 pp. Berlin, 1881.

TRANSACTIONS.

Berliner Astronomisches Jahrbuch für 1883, mit Ephemeriden der Planeten, 1881. 8vo, 508 pp. Berlin.

Monthly Notices of the Royal Astronomical Society. Vol. 41. London, 1881. Serial.

Proceedings of the U. S. Naval Institute. Vols. 1 to 5. Annapolis, 1875–80.

MISCELLANEOUS.

Buck, Albert H. A Treatise on Hygiene and Public Health. 2 vols. 8vo. New York, 1879.

Illinois Industrial University. Tenth Report of the Board of Trustees. 8vo, 261 pp. Springfield, 1881.

Petroff, Ivan. Population and Resources of Alaska. 8vo, 86 pp.

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GRADUATE DEPARTMENT.

OWEN FREDERICK OLMSTEAD.

We regret to learn, from the Secretary of the Class of '78, of the recent death of Owen Frederick Olmstead, at Albany, N. Y. A student of marked ability, and a man of rare promise, he was only prevented by ill health from rising to the front ranks of his chosen profession. Attacked with lung-trouble, he for some time struggled manfully against bodily weakness and pursued his studies at the school, completing the course, and receiving the degree of Civil Engineer. After graduating, he was compelled to seek health and strength in a more congenial climate, and on this account decided to abandon for the time the idea of practising his profession, and to accept a favorable opening near Cheyenne, Wyoming Territory, where, up to a short time before his death, he was engaged on a ranch, in stock-raising and agricultural work. Most favorable reports of improved health and increase of fortune have lately been received from him, and the news of his death was a surprise and shock, for which we were little prepared. His death, we understand, was due to consumption. A faithful, earnest student and friend, Mr. Olmstead will be missed by his classmates and all others who knew him.

The annual reunion of the Alumni Association of the School of Mines will take place on Thursday, December 29th, at the Westminster Hotel, corner of Sixteenth Street and Irving Place. It has been usual, heretofore, to celebrate the annual reunion in the early part of January, but, in order to accommodate graduates of the school who are obliged to return to their work at points distant from New York City, the date fixed for the dinner has been changed to December by the Board of Managers. We learn from the Resident Secretary, as we go to press, that this change has been productive of good results, there being every prospect of a large attendance and successful celebration.

A fresh supply of the Mining Engineering pins has just been received, and graduates can now obtain these badges by application to Professor Munroe, the member of the committee having charge of the same. The new Civil Engineering badge is in the hands of Dr. Hutton, and can be seen, and, we believe, ordered, from him. The design is neat and appropriate, and well carried out by the jeweller to whom was entrusted the making of the badge.

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GEORGE B. HANNA, E.M. ('68), has contributed an exceedingly valuable paper on the mines of North and South Carolina and Georgia, to the Report of the Director of the Mint just issued. We hope to be able to publish extracts from this report in the next number of THE QUARTERLY.

WILLIAM ALLEN SMITH, E.M. ('68), Manager of the Weldon Mining Co., has now an office in the new "Post Building," No. 16 Exchange Place, New York City, and proposes to make the examination and development of iron properties his special work, in which specialty his practical experience in New Jersey and elsewhere will be of great value.

EBEN E. OLCOTT, E.M. ('74), is, we learn, at present in Las Delicias, Sonora, Mexico, as Superintendent of the St. Helena Gold Mine.

SPENCER B. NEWBERRY, E.M., Ph.D. ('78), has returned from Germany, and is now, we learn, Assistant in Chemistry to Professor Caldwell, in the new Agricultural Department of Cornell University.

W. B. S. REED, E.M. ('79), is engaged at or near Elizabethtown, Hardin Co., Ky., on the work of location of the E. L. & B. S. R. R. He is anxious to know what the other members of '79 are doing.

F. T. ASCHMAN, Ph.B. ('81), is now Chemist to the Wheeler Iron Co., Pa. He writes as follows, under date of December 3, 1881: "Have just got to work, and think I shall make out all right. The laboratory here is very complete and nice, much more so than I expected, and since I am entirely my own 'boss,' I am very well satisfied. The work consists principally of analysis of iron ores, limestones, pig- and wrought-iron."

CHANDLER D. STARR, C.E. ('81), one of the original promoters of THE QUARTERLY, and during the first year its Editor-in-Chief, is now Resident Engineer on the E. R. R. R., with headquarters at Falls Mills, Tazewell Co., Va. Mr. Starr has had quite a valuable and varied experience since he left the school. On graduating, he accepted a position as transit-man on the N. Y., O. & W. R. R., in which capacity he served with credit till August, when the surveys of his division were completed. His present position in Virginia being offered at that time, he accepted it at once, preferring not to wait for a vacancy on the staff of the New York road, though he was assured that he should have the first that occurred. The first three months of his engagement in Virginia he occupied the position of transit-man, much of the time in charge of the party and locating the line; and when the construction of the road was begun, he was promoted to the position of Resident Engineer. He has now charge of the construction of over eleven miles of line, with seventeen bridges, and hopes for an extra mile or so, which will put two tunnels under his superintendence.

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We note that the Tombstone Mill and Mining Co., John A. Church, E.M. ('67), Superintendent, has just declared another dividend of $50,000; also the Chrysolite Silver Mining Co., Chas. M. Rolker, E.M. ('75), Superintendent, a mine a year ago supposed to be exhausted, a dividend of $100,000, furnishing tangible evidences of the sort most appreciated of the value of School of Mines graduates.

THE "School of Mines Register" for 1881-82 has just appeared. The names of the following graduates of the School appear in the list of officers: Munroe, '69; Waller, '70; Ricketts, '71; Colton, '73; Rees, '75; Wells, '75; Hutton, '76; Britton, '79; Beebe, '80; Churchill, '80; Greenleaf, '80; Torrey, '80.

The following men are also at the school as private assistants to professors or instructors: Colby, '77; Mackintosh, '77; Hathaway, '79; Hollick, '79; Wiechmann, '81.

The following are taking post-graduate courses of study: Neftel, '79; Braschi, '81; Leary, '81; and Williams, '81.

The number of students now in the school is 267, one man having joined the second class since we printed the last number of THE QUARTERLY.

THE American Chemical Society have just chosen their officers for 1882. The following graduates and former students of the school are included in the ticket elected: E. Waller, '70, as one of the Vice-Presidents; T. O'Connor Sloane, '72, Treasurer; W. Rupp, as one of the Curators; A. R. Ledoux, on the Committee on Papers and Publications; Arthur H. Elliott, '81, and A. P. Hallock, '80, on the Committee on Nominations; and P. de P. Ricketts, '71, on the Board of Directors.

THE following additional corrections to the graduate lists published in the last volume of THE QUARTERLY have been received. In the next number we shall print the revised list in full, and would, therefore, request all interested to send at once their own present address, or any other corrections which should be made, to the Resident Secretary of the Alumni Association.

1868—George Hampton Coursen, E.M., C.E., care G. H. Coursen, 307 Greenwich Street, New York City.

1868—William Pistor, E.M., Hamilton & Pistor, architects, 35 Broad Street, rooms 23 & 24; address, 211 West Forty-fourth Street, New York City.

1868—William Allen Smith, E.M., Mining Engineer, Post Building, 16 Exchange Place, New York City; address, Dover, N. J.

1870—Stuart Lindsley, E.M., care N. G. Lindsley, Orange, N. J.

1870—Theodore F. Van Wagenen, E.M., General Manager Twin Lakes Consolidated Placer Mining Co., Leadville, Colorado.

1872—Thomas O'Connor Sloane, A.M., E.M., Ph.D., 119 Pearl Street, Hanover Square, New York City.

1874—Eben Erskine Olcott, E.M., Superintendent St. Helena Gold Mine, Las Delicias, Sonora, Mexico.

1875—Philip Charles Pfister, E.M., 125½ Forsyth Street, New York City.

1876—Nathaniel Wright Lord, E.M., Professor of Engineering and Metallurgy, Ohio State University, Columbus, Ohio.

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1876—Frederick Furneux Hunt, E.M., C.E., The Huachuca Mining and Smelting Co., Pelton, Arizona.

1878—Willard Parker Butler, E.M., care Maitland, Phelps & Co., 22 Exchange Place, New York City.

1878—William Henry Smeaton, C.E., E.M., Mount Vernon, Westchester County, N. Y.

1878—Gouverneur William Morris, E.M., Myers, Rutherfurd & Co., 58 Wall Street, New York.

1878—Spencer Baird Newberry, E.M., Ph.D., Assistant in Chemistry, Cornell University, Ithaca, N. Y.

1878—Owen Frederick Olmstead, C.E., died at Albany, N. Y., November, 1881.

1878—Richard A. Parker, C.E., 347 West Forty-eighth Street, New York City.

1878—Nelson Williams Perry, E.M., 238 Auburn Avenue, Cincinnati, O.

1879—George Spencer Eastwick, C.E., Manager Sugar Refinery, care A. Thomson & Co., New Orleans, La., or care Havemeyer & Eastwick, Jersey City.

1879—Charles Arthur Hollick, Ph.B., School of Mines, New York City.

1879—Theophilus Smith Mathis, E.M., U.S. Deputy Mineral Surveyor, Ouray, Ouray Co., Colorado.

1879—George Fanshawe Milliken, E.M., Tucson, Arizona.

1879—Otis Mortimer Munroe, Ph.B., Ironton, Iron Co., Missouri.

1879—Knight Neftel, C.E., Neftel & Churchill, Civil and Mining Engineers, 39 Broadway, New York City.

1879—William Bell Stephen Reed, E.M., E. L. & B. S. R. R., Elizabethtown, Hardin Co., Ky.

1880—Nathaniel Butler, E.M., New York, West Shore & Buffalo R. R., 78 Carver Street, Boston, Mass.

1880—Robert Otto Francke, C.E., care D. O. Francke, Gothenburg, Sweden.

1880—Wilkins Updike Green, Ph.B., Mears' Chlorination Co., 434 Walnut Street, Philadelphia, Pa.

1880—Edward Henry Hudson, C.E., 54 East Fifty-third Street, New York City.

1880—Charles August Meissner, Ph.B., Chemist Briar Hill Iron & Coal Co., Youngstown, Ohio.

1880—Frederick Augustus Potts, Jr., Ph.B., 110 Broadway, New York.

1880—George Singer, E.M., Wilkinsburg, Alleghany Co., Pa.

1880—Maxwell Smith, C.E., New York, West Shore & Buffalo R. R.; address, 121 West Twenty-first Street, New York City.

1881—F. T. Aschman, Ph.B., Chemist Wheeler Iron Company, P. O. Box 292, West Middlesex, Mercer Co., Pa.

1881—H. Van F. Furman, E.M., 16 East Twenty-third Street, New York City.

1881—W. M. Meserole, C.E., New York, West Shore & Buffalo R. R.; address, 590 Lorimer Street, Brooklyn, N. Y.

1881—W. T. Richmond, Ph.B., Chemist U. S. Geological Survey, Newport, R. I.; address, 30 West Sixty-first Street, New York City.

1881—Chandler D. Starr, C.E., Resident Engineer E. R. R. R. Co., Falls Mills, Tazewell Co., Va.

1881—Hermann T. Vulté, Ph.B., Superintendent Columbia Chemical Works, 66 Water Street, Brooklyn, N. Y.

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THE

SCHOOL OF MINES QUARTERLY.

VOL. III. MARCH, 1882. No. 3.

THE PROGRESS OF THE METALLURGY OF GOLD AND SILVER IN THE UNITED STATES.

(*Continued.*)

BY T. EGLESTON, PH.D.

AMALGAMATION whether in the barrel or the pan was first used only for ores that are now called free-milling—that is, which will amalgamate immediately with the mercury. But it became evident soon that there was a large amount of ore which would not amalgamate. It contained sulphur and other substances which either prevented the action of the mercury altogether or caused a great loss of it. Such ores were called rebellious, and at first were not used; afterward they were roasted, the principal object of the roasting being not only to drive off the sulphur which was present, but by adding a little salt to convert the silver into chlorides which could be easily attacked by the mercury, which process was called the Reese River Process in distinction from the Washoe, which is the free-milling.

Up to this time stamping had generally been done wet in order to avoid the losses occasioned by the dust, as there was no reason why in the Washoe process the wet ore should not be delivered to the pans as soon as it settled sufficiently to form

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the "pulp," but with rebellious ores in regions where fuel was generally scarce, too much heat would have to be used in driving out the moisture from the ore,[1] and dry-stamping took the place of the wet. As the ore came from the mine damp, drying floors, made by running the flues of the roasting furnace backwards and forwards under the iron plates covering the floors behind the stamps, were used, and the ore from the mine was spread out over these plates until it was dry. As there was nothing but the force of the blow from the stamp-head to deliver the ore from the screens, the operation was slower, and as no water current carried the ore away from the front of the stamps, endless chains were placed in the dust-tight boxes which enclosed the front of the mortar-screens, which delivered the crushed ore into bins in the roof of the works, whence it was delivered by spouts into the furnace.

Every kind of furnace for roasting was invented and tried. These were usually some kind of reverberatory furnace, and were the subject of a large number of patents, and were altered and modified with more or less permanent success. Attempts were made to make the work wholly mechanical by the use of revolving cylinders into which the ore and salt were charged by machinery, in order to get rid of the difficult hand labor required, and the necessary exposure to fumes in roasting in ordinary reverberatory furnaces. A few of these survive in the Bruckner[2] and Teates furnaces, but as a general thing the cost of repairs to these contrivances, and the necessity of more engine-power or of separate engines in the absence of machine-shops near at hand, increased the expense of working beyond the gain in diminished labor.

On account of the large loss both in mercury and in silver which are carried off in the tails, mills[3] called tail-mills have been erected, where, notwithstanding the large loss in mercury which is known to exist in the treatment, there is a constant increase in the quantity of mercury, and very often a large yield in silver. These mills have no work to do unless the tails are accumulated by damming the valleys through which the tail-streams pass. When the accumulated tails have been treated, the mills have no further use, and very often the heavy freshets do the work first by sweeping the tails away. Sluices of great

[1] "The Stamp Mills of California," *Engineering*, London, Eng., vol. 31, p. 624.
[2] *Engineering*, vol. 22, p. 575.
[3] "Treatment of Tailings," *Engineering*, vol. 30, p. 395.

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length and width have been put up, but their tails are still rich.[1] Many machines have been invented for catching mercury and amalgam by making the tails pass over revolving blankets, rubbers, revolving amalgamated plates, and many other contrivances, but they are not as yet successful in the commercial sense.

When lead, copper, or zinc were present in the ores in any considerable quantity they became so rebellious that amalgamation was out of the question, and smelting, with its necessary adjunct of concentration, became necessary. As the dressed ores were rich and contained a large product condensed into a small one, and as this product was usually sold, sampling[2] works sprung up, in which the value of a large quantity of ore was carefully ascertained by processes more or less mechanical, in which, as rapidity of execution as well as correctness of results were necessary, a number of tools for reducing the ore to powder now generally used were invented.[3]

The first attempts at smelting, as they were usually conducted by persons of no great experience, were not very successful. In fact, the early history of the now very successful American methods is a record of failures. When the smelting of silver ores became a necessity, English methods were first introduced by the Cornish miners, only a few of the German and Swedish furnaces being used. But as the English type of furnace requires a considerable amount of good fuel, of a kind not generally found in the West, and the use of wood in them requires great skill, shaft furnaces gradually took their place, for the most part for treating ores containing gold, copper, silver, and lead by smelting.[4] Some of the processes adopted from the old works in Europe found themselves in circumstances where the conditions of transportation, labor, or fuel were such that they could not compete with other districts, so that they gradually disappeared, and were succeeded by the same processes in a new dress or in a new phase to such an extent that the plant and the process as it is now used in the West would hardly be recognized by their inventors. Little by little it was ascertained that when the ore contained any volatile material, although it might be in small quantities, it would carry off with it very considerable portions of the precious metal; and then arose the idea of condensing chambers, until gradually, without

[1] *Engineering*, vol. 30, p. 395.
[2] *Engineering*, vol. 22, p. 495.
[3] *Engineering*, vol. 22, p. 495, figs. 5 to 12.
[4] *Transactions of the American Institute of Mining Engineers*, vol. 4, p. 275.

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any one person having invented them, the methods have grown into the simple and very beautiful processes which are now in use in the West.

At first the only fuel used was charcoal, and this, as wood was scarce, was sometimes made from dead or from float wood, or from woods too light for the purpose, so that it very often happened that the charcoal would crush by its own weight, and would not stand a charge in the furnace at all. It then became evident that the ores must have a comparatively high yield, and that as they usually had a gangue composed for the most part of silica, coke was necessary, and this, with from ten to twenty per cent of ash,[1] was imported from the East or from Europe at an expense in some of the works of forty dollars a ton. As it became evident that coke was the necessary fuel, it also became apparent that something must be done to reduce its cost, and also the cost of the refractory materials. Water-back furnaces were then introduced, which consisted of cast or sheet iron boxes riveted together, surrounding the hottest part or the whole of the furnace and cooled with water. These furnaces were for the most part open-breast furnaces with front hearths, and were continually getting out of order from the formation of sows and bears which occasionally stuck to the bottom or formed engorgements in the furnace higher up. To avoid these the Ahrent's tap was invented, which keeps a very considerable quantity of lead on the hearth of the furnace at all times, and allows of the casting being done without interfering with the interior. In other works where copper is in large quantity the ores are smelted for copper, and the silver and gold concentrated in one of the products, from which it is separated in the wet way by the skilful adaptation of old processes.[2]

It now became evident that there were considerable mechanical losses from the metal being carried up and out of the chimney, so that in some instances in Utah as much as ten or even fifteen or twenty per cent was carried off in this way. Attempts were then made to collect this material as is done in Europe in condensation chambers of large size and extent, and several systems of doing it have been invented. The simplest, the cheapest, and the most recent of these is used at Mansfield Valley, near Pittsburgh, Pennsylvania,[3] and is an adaptation of the flue

[1] The high percentage of ash in the coke has in several instances caused the failure of works which with a suitable fuel might have been successful.

[2] *Trans. Amer. Inst. Min. Engs.*, vol. 4, p. 276.

[3] *Annals N. Y. Acad. Sci.*, vol. 2, p. 106.

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leading to the chimney by dividing it in two sections vertically. In the lower one of these, partitions eighteen inches high, placed four or five feet apart and one third of the height of the flue, catch the dust by gravity, and as there is no velocity below, it remains there. The gas circulates above.

Generally the silver and the gold in a district where lead ores can be had, are concentrated in a pig lead improperly called "base bullion." In some few cases in the early days the German method of cupellation was used, but as this requires a maximum consumption of fuel, great skill, and a market for litharge, it was quickly superseded by the English method, which requires less skill, makes no litharge for sale, but required the poor lead to be concentrated into a rich one and that treated for gold and silver. The Patterson process by crystallization for enriching the lead previous to cupellation was never extensively used here, principally because at the time when there was a large amount of work to be done the process had already been superseded. The lead directly from the furnace is now enriched by zinc desilverization,[1] and the rich lead cupelled in an English furnace.

The history of this process is very peculiar. Invented in 1842 by Karsten, it was declared a failure after a prolonged investigation by that very able metallurgist. It was reinvented by Crooks in 1858 in England, where it was not very successful, and was brought to this country as an English process. It was tried again at Tarnowitz, in Silesia, and was more or less of a failure there, and was then reintroduced into this country, and so many improvements made in it that to-day the American modification of it has become the perfection of a process, and the furnace used a type furnace.

In order to use the method of desilverization by zinc it is necessary that the lead should be very pure. To purify the lead small refining furnaces were used in Germany, containing two to three, and subsequently from five to six, tons each. But in this country one of the first improvements made was the softening or purification of the lead in a furnace containing from fifteen to twenty, and subsequently as high as twenty-six tons; but as the hearth of such a furnace was difficult of construction, it was simply made in a cast or wrought iron pan. This softened lead had to be discharged from the furnace, which was not an easy matter, and the late Mr. Steitz

[1] *Annals N. Y. Acad. Sci.*, vol. 2, p. 86.

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invented a siphon[1] to do it, which seemed to be the perfection of an instrument for this purpose.

The refined lead is stirred with zinc, the zinc-scums carrying the silver with them are liquated to separate the excess of lead, and the result is a very rich zinc alloy, containing a large amount of lead, which is granulated and distilled in retorts. The distillation in retorts promised at one time to wreck the process, as it had to be effected in small furnaces[2] surrounded by coke, and the number of retorts broken was large, notwithstanding the use of Steitz's siphon. Petroleum was then tried with great success lessening the breakage of the retorts due to the charging of the fuel and the poking of the fire. Subsequently Mr. Faber du Faur invented his tilting furnace, which allows of pouring the rich silver lead out of the retort without disturbing it, thus removing all the difficulty. The silver lead from which the zinc has been distilled is cupelled in an English furnace and cast into pigs. The lead from which the silver has been removed is refined in a furnace similar to the softening furnace, called a calciner. All the lead so refined is of the highest quality fit for the manufacture of white lead. It is produced almost as a by-product, and at a low cost.

The improvements in cupellation have been, first, the invention of the iron cupel surrounded by water, by the late Mr. Steitz, of St. Louis, upon which the lead could be brought up to fine silver, and the later invention of Mr. Eurich, of the Pennsylvania Lead Company, of going from the lead riches to silver 996 fine, on a hearth made of Portland cement, and casting directly from the cupel into silver bricks by a simple arrangement for tipping the cupel.[3]

It sometimes happens that silver can be extracted from its ore in the wet way. There are three principal methods which have been used for this purpose. The first was introduced in 1849 by a German named Augustine, and consists[4] in transforming the ore into chloride by roasting the ore to drive off the sulphur and other impurities, grinding the roasted ore and then roasting with salt to form chlorides; then dissolving out the chlorides with a saturated solution of salt, precipitating the silver with copper, compressing and melting the silver. It is usual to concentrate the silver into copper mattes for this pro-

[1] *Annals N. Y. Acad. Sci.*, vol. 2, plate 7, fig. 6; plate 9, fig. 10.
[2] Ibid., p. 98. [3] Ibid., p. 108.
[4] *Trans. Amer. Inst. Min. Engs.*, vol. 4, p. 295.

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cess. Shortly after the invention of this process, another, much simpler, was invented by Ziervogel,[1] which consists in roasting mattes to produce sulphates, decomposing all these sulphates except that of silver, and then dissolving out the sulphate of silver with hot water. Simple as it appears, this process is exceedingly difficult to execute, for it requires a very high degree of skill to seize the exact moment when all the sulphates of the other metals are decomposed and none of the silver is. If the sulphates are not all decomposed the silver is precipitated by them; if they are, there is danger that the silver sulphate also will be decomposed, and it will then be lost, as the oxide is not soluble. The practice, therefore, is to leave a little of the sulphates of these metals undecomposed, as the loss in this case can be calculated beforehand, while no one can tell what it will be in the case of too much roasting. As these residues are always rich, they are often treated by the Augustine process, the two being very advantageously used together in this country.[2]

In looking at the silver process as a whole and comparing the cost, we find that the relations between the relative cost and quantity of silver extracted were very interesting.

	Relative Cost.	Relative Loss.
Amalgamation	2.2	2.0
Augustine process	1.8	2.0
Ziervogel "	1.0	1.0

The Ziervogel process is, both as to cost and residues, twice as advantageous as the others.

Still another process was invented in 1858 by an Austrian of the name of Von Patera, which consists in roasting, as in the Augustine process, leaching with hot water before roasting with salt in order to dissolve out any soluble salts, roasting with salt, then dissolving the chlorides with hyposulphite of soda, precipitating the silver with polysulphite of sodium, and then reducing the sulphide of silver. This process is easily carried out, in that the reagents can readily be had, and that none of them are wasted; but both the lixiviation and the precipitation require such nice distinctions and such an exact chemical knowledge that it has not been very successful.

The bullion which is produced as the result of treatment of any of the ores usually contains some small quantity of the base metals besides the gold and silver. The gold from Cali-

[1] *Trans. Amer. Inst. Min. Engs.*, vol. 4, p. 245. [2] Ibid., p. 295.

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fornia generally contains about twelve per cent of silver, that from Australia four to six per cent. The amount varies from three per cent to about twenty-five per cent. The silver bullion often contains gold, as in the case of the Comstock, where one third of its value is gold; and these metals must be separated in order that they may be alloyed to their proper standards for commercial uses. Neither pure gold nor pure silver is of any use commercially except for electroplating; for all other purposes they would be much too soft. The process of separation is called parting. To effect this an alloy is made by melting, which usually contains three parts of silver to one of gold. In a single instance in California this alloy is three of silver to two of gold. The formation of this alloy is called quartation or inquartation. It is granulated and subjected to one of three different processes; the silver is dissolved out by either nitric or sulphuric acid, and in both cases the residue not dissolved will be gold. The nitrate of silver siphoned off from this is diluted with water and precipitated with salt. The chloride of silver so formed is reduced to a metallic state with sulphuric acid and zinc, and the silver melted into bars whose fineness is stamped on them, and they are then used for commercial purposes. In the case of sulphuric acid, sulphate of silver is formed, which is diluted with hot water and precipitated as metallic silver by copper. The spongy silver is pressed into cakes by a hydraulic press and melted into bars. The gold is collected, melted, and run into bars. The nitric acid method has been generally abandoned, because it poisons the neighborhood with fumes. The sulphuric-acid process, which is a little cheaper, has taken its place, except in California, where a very beautiful method, invented by Mr. Gutzkow,[1] has taken its place. This method, which is very ingenious, much quicker and gives better results than the others, was introduced in San Francisco in 1867. Most of the alloys are not granulated; they are inquartated and dissolved in sulphuric acid, in bars. The sulphate of silver is crystallized and a solution of sulphate of protoxide of iron run through it, which reduces the silver to a metallic state; the iron solution becomes sulphate of sesquioxide of iron, and is restored to its original condition with fresh iron and used again. The silver is pressed and melted as before, and the gold from the pots treated in the same way. This pro-

[1] Burchard, "Production of Gold and Silver." Washington, 1881, p. 356.

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cess is not only very simple, but is in a chemical way one of the most beautiful known.

The amount of silver produced in the United States previous to 1858 was so insignificant that no statistics have been recorded. In that year it was only $500,000; in 1859 it was only $100,000; in 1860 it was $150,000; but in the following year, 1861, the amount of silver began gradually to increase, until the year 1870, when it was $16,000,000. The total amount produced in this decade from 1860 to 1870 was $84,300,000, the lowest amount being $150,000, in 1860. From 1870 to 1880 the amount of silver becomes a very considerable factor in the world's production of this metal, the highest amount being a little over $45,000,000, in 1878, and the lowest $16,000,000, in 1870, the total production for the decade being $374,922,260. If these amounts of silver are added to the amounts of gold shown in the table, in the last number of the QUARTERLY, p. 80, it will be seen that for the decade from 1860 to 1870, the highest production of precious metals was in 1869, $61,500,000, and the lowest was $43,700,000, in 1862, the total being $555,150,000. For the next decade, from 1870 to 1880, the highest production was in the year 1878, $96,487,745, and the lowest $66,000,000, in 1878. The largest amount of silver produced by any one State during the year 1880 was $17,000,000, obtained in Colorado, and the next largest $10,000,000, from Nevada.

YEAR.	Silver.	Total Gold and Silver.	YEAR.	Silver.	Total Gold and Silver.
1858.....	$500,000	$50,500,000	1870.....	$16,000,000	$66,000,000
1859.....	100,000	50,100,000	1871.....	23,000,000	66,500,000
1860.....	150,000	46,150,000	1872.....	28,750,000	64,750,000
1861.....	2,000,000	45,000,000	1873.....	35,750,000	71,750,000
1862.....	4,500,000	43,700,000	1874.....	37,324,594	70,815,496
1863.....	8,500,000	48,500,000	1875.....	31,727,560	65,195,416
1864.....	11,000,000	57,100,000	1876.....	38,783,016	78,712,182
1865.....	11,250,000	64,475,000	1877.....	39,793,573	86,690,963
1866.....	10,000,000	63,500,000	1878.....	45,281,385	96,487,745
1867.....	13,500,000	65,225,000	1879.....	40,812,132	79,711,990
1868.....	12,000,000	60,000,000	1880.....	37,700,000	73,700,000
1869.....	12,000,000	61,500,000			

Such enormous productions of the precious metals have not been without their influence on the relative value of gold and silver in other countries. The United States is one of the largest producers of the precious metals, notwithstanding, as the statistics show, there has been a gradual falling off in the production of gold, and the highest limit of silver appears to have been in

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the year 1878, since which time the decrease in the production of the Comstock has brought down the production of silver from its maximum in 1878, nearly $8,000,000, and it seems likely that this decrease will continue.

The amount of gold consumed in the United States for purposes of art and ornament during the year 1880 was larger than for several previous years. The following table from the Report of the Director of the Mint, which is a mine of information for those interested in the production and distribution of the precious metals, gives the returns of the New York Assay Office for that year:

BARS MANUFACTURED.[1]

	Gold.	Silver.	Total.
From United States coin (defaced)	$4,929	$982	$5,911
Foreign coin	260,222	72,668	332,890
" bullion	1,007,400	278,622	1,286,022
Domestic "	2,988,422	3,863,126	6,851,548
Plate, etc	394,871	144,992	539,863
Total	$4,655,844	$4,360,390	$9,016,234

From the whole United States this amount is much larger; but leaving out the foreign bullion altogether, the following table gives the estimate of the total gold and silver used in the whole United States for industrial purposes during the last year:

	Silver.	Gold.
Domestic bullion	$4,000,000	$5,000,000
U. S. coin	600,000	2,500,000
Plate, foreign bullion, and coin	400,000	2,500,000
Amount consumed	$5,000,000	$10,000,000

The consumption of the precious metals for purposes of art and ornament has been the subject of estimates by many distinguished statisticians, but at the best can only be approximated. In 1827 Humboldt placed it at 375,000 ounces, or one fifth of the world's production at that time. In 1822 Lowe estimated it at two thirds. William Jacob estimated it at 988,000 ounces, which was double the average annual production between 1821 and 1830. Dr. Soetbeer, of Germany, gives the fol-

[1] Report Director of the Mint, 1880, p. 19.

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lowing tables of the consumption of the precious metals for jewelry and other industrial purposes in the various countries of the world:[1]

GOLD.

	Consumption, in Ounces.	Reduction by Old Material used.	Total Consumption.
United States	529,000	10 per cent	476,000
Great Britain	703,000	15 "	598,000
France	739,000	20 "	591,000
Germany	518,000	20 "	412,000
Switzerland	529,000	25 "	397,000
Austria	102,000	15 "	87,000
Italy	212,000	25 "	159,000
Russia	106,000	20 "	85,000
Other Countries	176,000	20 "	141,000
Total	3,614,000		2,946,000

SILVER.

	Consumption, in Ounces.	Reduction by Old Material used.	Total Consumption.
United States	4,233,000	15 per cent	3,789,000
Great Britain	3,175,000	20 "	2,540,000
France	3,528,000	25 "	2,646,000
Austria-Hungary	1,411,000	20 "	1,129,000
Switzerland	1,129,000	25 "	847,000
Italy	882,000	25 "	662,000
Russia	1,411,000	20 "	1,129,000
Germany	3,528,000	25 "	2,646,000
Prussia	1,870,000		1,411,000
Total	21,167,000		16,799,000

Other estimates give the entire consumption of the precious metals in Europe and America for industrial purposes in 1880 as from $45,000,000 to $55,000,000 in gold, and from $25,000,000 to $30,000,000 in silver.

From 1831 to 1880 the estimated consumption of gold for industrial purposes was 73,000,000 ounces, or 32.6 per cent of that produced. For silver it was 511,000,000, or 25.2 per cent.

Of the world's product of bullion it is estimated that one third is used up and lost in the wear and tear of coins and articles made for use or ornament, one third is used for manufacturing purposes, and one third goes to supply the increased demands of trade. The amount lost by the abrasion of coins is shown by the fact that the average life of an English sovereign

[1] *Engineering and Mining Journal*, vol. 32, p. 183.

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is eighteen years, by which time the coin has lost three quarters of a grain and is no longer legal tender. Dr. Soetbeer[1] states that the annual loss from this source in civilized countries reaches 28,000 ounces of gold and 1,600,000 ounces of silver.

The following table[2] shows the amount of gold and silver produced in the world in the years 1877, 1878 and 1879:

	1877.		1878.		1879.	
	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
United States	$47,897,390	$39,793,573	$51,206,360	$45,281,385	$38,899,858	$40,812,132
Russia	27,226,668	467,844	27,967,697	448,016	26,584,000	415,676
Australia	29,018,223	.....	29,018,223	.....	29,018,223	.....
Mexico	996,898	27,018,980	996,898	27,018,940	989,161	25,167,763
Germany	204,697	6,135,877	205,361	6,938,073	205,361	6,938,073
Austria	1,196,278	2,119,948	1,196,278	2,161,515	1,062,031	2,002,727
Sweden	2,658	54,038	6,001	52,708	1,994	62,435
Norway	.....	188,052	.....	166,270	.....	166,270
Italy	72,375	17,949	72,375	17,949	72,375	17,949
Rest of Europe	.....	2,078,380	.....	2,078,380	.....	2,078,380
Argentine Republic	78,546	420,225	78,546	420,225	78,546	420,225
Colombia	4,000,000	1,000,000	4,000,000	1,000,000	4,000,000	1,000,000
Rest of S. America	1,993,800	1,039,190	1,993,800	1,039,190	1,993,800	1,039,190
Japan	265,840	706,649	295,746	728,846	466,548	916,400
Africa	1,993,800	.....	1,993,800	.....	1,993,800	.....
Total	$113,947,173	$81,040,665	$119,031,085	$87,351,497	$105,365,697	$81,031,220

Dr. Soetbeer[3] gives the totals as:

1877.		1878.		1879.	
Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
$121,514,026	$96,855,376	$122,058,368	$104,126,608	$104,245,987	$102,229,521

These tables show that the United States is by far the greatest producer of the precious metals, Russia being the only one which produces anything like as much gold, and Mexico the only one that approaches it in silver.

The amount of precious[4] metals sent to the East, the greater part of which goes to India, has been estimated by Dr. Soetbeer as:

	Gold, in Ounces.	Silver, in Ounces.
1831–1840	35,000	7,750,000
1871–1880	423,000	38,000,000
1831–1880	19,700,000	1,376,000,000

In the period from 1871–1880, which is most reliable, the consumption of gold by this means was 47,000 ounces, and of silver 4,200,000 ounces. In India alone the imports in the last forty years have exceeded the exports of these metals by $400,000,000, of which only $8,000,000 have been coined as money.

[1] *Engineering and Mining Journal*, vol. 32, p. 182.
[2] Report Director of the U. S. Mint, 1880, p. 159.
[3] *Engineering and Mining Journal*, vol. 32, p. 182.
[4] Ibid., p. 183.

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The amount of the precious metals hoarded or put out of circulation either as objects of art or ornament is becoming greater with every decade. It appears from the data given page 167 that the total annual consumption of the precious metals for purposes other than coinage is about 3,600,000 ounces of gold and 21,000,000 of silver. It has been estimated that the entire amount of gold now in the world is only equal to that which has been produced in the last twenty-five years, and that of silver to that produced in the last eighty years. No one has as yet been able to satisfactorily explain what has become of all the rest of the precious metals.

Only an estimate can be made of their wear and tear, which is an irretrievable loss, either in the abrasion of coin or in the use of leaf or of the pure metals for plating purposes. Add to this the amount lost in lead, copper, and other metals, which do not contain enough of it to separate, and it is not a matter of surprise that, notwithstanding the enormous yearly increase, the estimate of the total amount supposed to exist in the world during any period is not perceptibly greater.

In all the methods for the extraction of the precious metals there are considerable losses. With the perfection of processes, the main object is to reduce them, or else to cheapen the labor of extracting the ores. These losses are greater than is usually supposed, because as a general rule systematic assays of the tails are not made. Yet it is known that the tails contain precious metals, and they are sometimes re-worked with profit, especially those from the silver mines. An interesting investigation was made some years ago, the results of which are given below,[1] showing the great loss in some of the mills.

	Yield of Ore in the Mill.	Remaining in Tails.		Total in Tails.
		Gold.	Silver.	
At the mill.....	$18 60	$10 04	$3 14	$13 18
Same tailings 350 ft. from mill....	18 60	5 02	3 92	8 98
Average yield of 150 tons.....	3 50	13 55	6 28	18 83
" " "	3 50	8 79	6 28	15 07
Slime from end of sluice 310 ft. long.....		56 00	33 30	89 00

It was also found that water from the mills three fourths of a mile below them contained in suspension, as an average of

[1] Report of the U. S. Mining Commissioner, 1872, p. 17.

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twelve assays, $0.018 per gallon. There were in this locality 576,000 gallons of this water flowing away in twenty-four hours, or a loss of $339.84. It was estimated that the annual loss of two mills working 250 days in the year was $84,960. From these and similar data the conclusion is drawn, that the loss is between fifty and sixty per cent of the total yield of the ore.

It is a matter of great interest to ascertain what the cause of these losses is, in order to learn how far they are capable of remedy. The first of these is undoubtedly a desire to get the largest possible output from the mill. This makes the ore too coarse to have all the gold and silver amalgamate, as part of it may not be released from the gangue. It would be much better to get the output by a more careful sizing of the ore, not forcing the stamp to do the work of a Blake's Crusher, and not sending to the mortars any ore fine enough to pass the screens. This is a matter of some importance, for it has been found with all kinds of stamps using screens, that it takes just as long to get crushed ore which has already passed the screens out of the mortar, as it does to crush and force it out. Too fine crushing is also quite as bad, for it produces "float," and is quite likely to put the precious metals in a condition in which they will not amalgamate.

Supposing that the losses which result from improper working do not exist, there are a few causes of loss which do not always amount to much, but which, in the early days, were a source of considerable loss. It has been found that any holes in the castings of the stamps, pans, etc., will attract the amalgam, and that it will even be carried into holes deep in the interior of the piece. This was a source of profit in the early days to those who recovered the precious metals when the worn out castings were melted. Another loss may be in cleaning the plates by taking off the amalgam too thoroughly. It is a well-known fact that new plates do not act as readily as old ones; the difference is so great, that when the mills can afford it, new plates are coated with gold or silver amalgam. Gold and silver will go much quicker to amalgam than to mercury. Too slow a current of water will keep the surface of the plates covered with a film of sand; a too rapid current will prevent the gold from being caught. If the gold is attached to a piece of the gangue rock which is relatively large, the specific gravity may be so reduced as to prevent the particles from coming in contact with

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the mercury. If the blankets are left too long without washing, so that the hairs become charged, the fine particles of gold are lost. If all these causes of loss are avoided, there are still others. For if the mercury is not kept clean or made so by chemicals, the "quick," having an extremely thin film upon it, does not act upon the gold or silver. Exactly the same effect is produced to a small extent when the rock is soapy, as is the case with some of the magnesian and aluminous rocks. If there are too few amalgamating machines, if the sluices are too short, there is also a loss. A very important source of loss is the flouring of the mercury from too rapid motion, or from the too free use of chemicals. In such cases steam may be used, if it is live steam fresh from the boiler, which prevents flouring by the expansion of the globules. If steam from the engine is used as a matter of economy, it often increases the loss, as very minute particles of grease are always carried off with it, which coat the mercury. The cause of the losses on the concentrates has already been discussed.

DETERMINATION OF HEATING SURFACE REQUIRED IN VENTILATING FLUES.

By Prof. W. P. Trowbridge.

The proper ventilation of a room requires the frequent removal of the vitiated air and the introduction of fresh air, the quantities by weight of the air ejected and the air introduced being equal. If the process be continuous and the proper amount of air be admitted and removed every hour, or minute, the only other requirements are that the entering air shall be pure, that it shall be properly warmed in cold weather either before it enters the room or by the mixture of warm and cold air in the room, and that the introduction and removal of air shall take place by gentle or inappreciable currents distributed in such a manner that the pure air may be thoroughly diffused throughout the room before it is removed.

These simple rules are easily stated and comprehended. It is also well understood that the movement of air requires force

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in proportion to the mass moved and the velocity which is imparted to it. The problems which arise in ventilation consist mainly in determining the position, arrangement, and size of the passage through which the air enters and leaves the space ventilated, and the adaptation to the passages or flues of the force or forces which produce the movement of the air. On the solution of these problems, too often misapplied or misunderstood, successful ventilation depends.

The various means of producing the movement of the air are

1st. *Fans or blowers*, requiring the use of machinery in driving them.

2d. *Ventilating chimneys*, in which the movement of the air is caused by the difference in weight between the column of hot air in the flue and the cooler air outside. This requires necessarily that the air in the flue shall be warmer than that outside, and the necessary heat must be imparted to the air before it enters the flue.

3d. The movement of the air may be produced by *steam jets* in the flue, the particles of steam issuing at a high velocity, carrying with them particles of air and thus producing a diminution of pressure below the jet and a consequent movement.

The steam jet method is seldom used, the ventilating flues being generally employed. The most common application of the latter is the ordinary ventilating flue of dwelling houses. In this case reliance is usually placed upon the heated air of the room, which, finding its way into the flue, forms a vertical column lighter than the outer air and having a height equal to the height of the flues. An upward movement in the flue is thus produced.

The laws of the movement, involving the height of the flues, the temperature of the air in the room, the temperature of the outer air, and the velocity of the upward current, may in this case be theoretically determined; and from such a determination, taking into account the frictional resistances, the area of the cross-section of a flue for the discharge of a given weight of air may be ascertained. Analytical investigations of this character are seldom made except by experts, and for ordinary dwelling houses the whole system is generally a haphazard construction in which the errors are quite certain to fall on the side of inefficient ventilation. Moreover the system cannot operate

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during those periods of the year in which the temperature of the outer air is the same or nearly the same as that of the room. During these seasons some other means must be resorted to in order to warm the air before it enters the vertical ventilating flues. The system is inefficient also, except with very large flues, owing to the slight difference, even in ordinary cold weather, between the temperature of the air in the room and that of the air outside.

For very active ventilation, therefore, with such flues as may ordinarily be constructed, the air must be heated; and this may be done by several methods:

1st. By a stove at the base of the flue, the stove-chimney passing up through the flue.

2d. By gas-jets burning at the base.

3d. By coils of steam-pipes in a chamber at the base of the flue.

All of these processes require careful analysis in their application, and may, to a certain extent, be subjected to such an analysis. The object of this paper is to investigate the laws which govern the ventilation when the air is heated by means of coils of steam-pipe, before it enters the flue. These laws have not heretofore been developed, and as this system of ventilation is a very simple one, capable of extended application, it is hoped that the following analysis may at least lead to a full discussion of the subject.

Let it be supposed that the air in the room is to be renewed at the rate of W pounds per second. The volume of this quantity of air can easily be determined when its temperature is given. Suppose also that it is to be ejected through a vertical flue whose cross-section is A and whose height is H, and that it is to be heated by steam-coils at the base of the flue, having a total exterior surface equal to S.

The following notation will be used:

W = weight of air discharged per second.

H = height of flue.

S = the exterior surface of steam-coil.

A = area of cross-section of flue.

T_a = the absolute temperature of the external air, *i.e.*, the common temperature Fahrenheit + 459.4°.

T_c = the absolute temperature of the air in the flue.

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T_s = the temperature of the steam in the coils.
D_a = the weight of a cubic foot of the external air.
D_c = the weight of a cubic foot of the air in the flue.
V' = the theoretical velocity of the air in the flue.
V = its actual velocity when frictional resistances are taken into account.
r = the rate in units of heat per hour, for each square foot of the surface S, at which the air receives heat from the coil, and for each degree of difference between the temperature of the steam and the air of the room.
K = a coefficient of velocity, such that $KV = V'$.
p = the unbalanced upward pressure at the base of the flue due to the difference between D_a and D_c, or due to the difference in weight between the two volumes of the height H within and outside of the flue.

The pressure per square foot, p, will then be represented by the equation,

$$p = HD_a - HD_c \text{ or } p = H(D_a - D_c) \qquad (1)$$

This pressure may be represented by the weight of a column of the heated air, represented by

$$\frac{p}{D_c} = H\left(\frac{D_a - D_c}{D_c}\right) \qquad (2)$$

The velocity with which the air would flow through the flue if there were no resistances would be determined by the expression,

$$\frac{V'^2}{2g} = H\left(\frac{D_a - D_c}{D_c}\right)$$

$$V' = \sqrt{2gH\left(\frac{D_a - D_c}{D_c}\right)} \qquad (3)$$

But, from the Marcotte-Gay Lussac law for air,

$$\frac{D_c}{D_a} = \frac{T_a}{T_c} \text{ or } D_c = D_a\left(\frac{T_a}{T_c}\right)$$

Substituting this value of D_c in equation (3), there results

$$V' = \sqrt{2gH\left(\frac{T_c - T_a}{T_a}\right)} \qquad (4)$$

In this expression the theoretical velocity is expressed in terms of the absolute temperatures of the air within and without the flue, and the height of the flue.

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From (4) we obtain

$$T_c - T_a = \frac{V'^2}{2gH} \cdot T_a \qquad (5)$$

The quantity of heat transferred to the air on its passage through or among the steam-pipes may be represented by

$$Q = WC(T_c - T_a) \qquad (6)$$

in which $C = .238$, the specific heat of air at constant pressure.

All of the above formulas are well known; the following are believed to be new:

The quantity of heat imparted per second to the air by the steam-pipes may be represented by the total exterior surface of the steam-pipes (S) expressed in square feet, multiplied by the rate per second and per square foot at which heat is transferred to the air, or

$$\frac{Sr(T_s - T_a)}{3600} = Q'$$

and from the nature of the problem we must have

$$Q = Q'$$

or

$$\frac{Sr(T_s - T_a)}{3600} = WC(T_c - T_a) \qquad (7)$$

from which we obtain

$$T_c - T_a = \frac{Sr(T_s - T_a)}{WC \times 3600} \qquad (8)$$

Combining this equation with equation (5), we have

$$\frac{Sr(T_s - T_a)}{WC \times 3600} = \frac{V'^2}{2gH} \times T_a$$

and

$$S = \frac{V'^2}{2gH} \times \frac{WCT_a}{\frac{r(T_s - T_a)}{3600}} \qquad (9)$$

This expression gives the total heating surface of the steam-pipes in terms of the velocity, the height of the flue, the weight of air discharged per second, and the absolute temperature of the external air.

If we substitute for V' its value KV, the expression becomes

$$S = \frac{K^2 V^2 \times WCT_a}{2gH\left(\frac{r(T_s - T_a)}{3600}\right)} \qquad (10)$$

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and since V is the actual velocity, we have

$$W = D_c V A$$

$$\therefore S = \frac{K^2 V^2 \times D_c C T_a A}{2 g H \left(\frac{r (T_s - T_a)}{3600}\right)} \qquad (11)$$

A being the area or cross-section of the flue. Equations (10) and (11) represent the laws connecting the heating surface with the height of the chimney, the area of the flue, and the temperature of the external air. They show that the heating surface is directly proportional to the cube of the velocity, to the area of the flue, and to the temperature of the external air, and inversely proportional to the height of the chimney and the rate of transfer of the heat, r, or $r\,(T_s - T_a)$.

To illustrate the practical use of the formula, let it be proposed to renew the air of a room containing 50,000 cubic feet four times in an hour, through a flue whose cross-section $A =$ 12 square feet, the height (H) being 50 feet. Required the number of square feet in the surface of the coil, the steam within the coil being maintained at 50 pounds pressure. Suppose, further, that the temperature of the air, before it enters the flue, is 60° F., the same as that of the outside air.

The weight of a cubic foot of air at 60° F., or (D_a), is found as follows: The weight of a cubic foot of air at 32° F., and atmospheric pressure being known, viz., 0.0808 pound, the weight at 60° F. will be

$$D_a = 0.0808 \frac{T_o}{T_a}$$

T_o being the absolute temperature corresponding to 32° F. (or $32° + 459.4° = 491.4°$), and T_a being equal to $60° + 459.4° = 519.4°$,

$$\therefore D_a = 0.0808 \frac{491.4}{519.4} = 0.0764 \text{ lbs.}$$

and the weight of 50,000 cubic feet at 60° F. will be

$$50000 \times 0.0764 = 3820 \text{ lbs.}$$

As four times this weight is to pass through the flue in one hour, we have

3,820 × 4 = 15280 lbs per hour;
or 254.67 lbs. per minute;
or 4.244 lbs per second.
$\therefore W = 4.244$ lbs.

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We also have

$$T_a = 519.4^\circ$$
$$C = 0.238$$

For the value of r to be used in the formula, an average of the results of the experiments made by Mr. C. B. Richards, at the Colt's Arms Co. at Hartford, may be taken for ordinary forms of heaters, it being understood that this value is assumed to be nearly correct, although the quantity (r) was found to vary in proportion to the quantity of air passing through the heater in a given time. Assuming the value of (r) to be 2.5 units of heat per hour per square foot of surface, and for each degree difference of temperature between the steam in the coil and the air of the room, there results for the value of r per second,

$$\frac{r}{3600} \times (T_s - T_a)$$

With steam at 50 pounds,

$$T_s - T_a = 281^\circ - 60^\circ = 221^\circ \text{ F.}$$

$$\therefore \frac{r}{3600}(T_s - T_a) = \frac{2.5}{3600} \times 221 = 0.153$$

All the quantities in equation (10) are now known for this problem, except K^2V^2. If we assume a velocity V, and also a correct value of K, the resultant value of S should be such as to give this velocity. Assuming $V = 7$ feet per second, and $K = 4$, the value of K^2V^2 will be $16 \times 49 = 784$.

In regard to the value of K, Peclet found in the ordinary boiler chimney of manufacturing establishments, for heights of 10, 20, and 30 metres, V' equal respectively to $5.6V$, $6V$, and $6.3V$. It may also be demonstrated that if the resistances were due to friction alone, the loss of velocity in a simple rectangular flue, 50 feet high and 12 square feet cross-section, would be such that V' would be about twice V, or $V' = 2V$.

The interposition of the coil of pipe through which the air passes increases the resistance beyond that of the simple flue, but it cannot be so great as that developed in passing through the grate, flues, and chimney of a steam-boiler.

I have taken for this example $V' = 4V$, *i.e.*, the actual velocity equal only to one-fourth of the theoretical, which will at least throw the error, if there be any, on the side of excess in the value of S.

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We may now recapitulate the value of the quantities in equation (10) as determined for this problem.

$K^2V^2=784.$

$\frac{r(T_s-T_a)}{3600}=0.153.$

$W=4.244.$

$C=0.238.$

$T_a=519.4°.$

$\therefore S=\frac{K^2V^2\times WCT_a}{2gH.\frac{r(T_s-T_a)}{3600}}=835$ square feet.

It now remains to ascertain what temperature and density the air in the flue will have.

The quantity of heat expended per second will be:

$S.\frac{r(T_s-T_a)}{3600}=S\times 0.153=835\times 0.153=127.755$ units. Per minute it will be 7,665.30, and per hour, 459,918 units.

To find the flue temperature (T_c) we have

$Q=WC(T_c-T_a)$

or $T_c=\frac{Q+WCT_a}{WC}=\frac{127.755+1.01\times 519.4}{1.01}=645.9°$, corresponding to a thermometric temperature of $645.9-459.4=186.5°$. This is the temperature of the air in the flue. Its density will be

$D_c=0.0808\frac{T_o}{645.9}=0.0808\frac{491.4}{645.9}=0.0761$ lbs. per cubic foot.

The volume of one pound will be $\frac{1}{0.0761}=13.14$ cubic feet.

With a velocity of 7 feet per second there will be ejected through the flue of 12 square feet area, 302400 cubic feet, or $302400\times 0.0761=23004.64$ pounds per hour.

This is in excess of the requirement, which was only 15280 pounds per hour.

If we assume a velocity of 5 feet per second, the discharge of heated air will be $12\times 5\times 3600=216000$ cubic feet per hour. The resistance to the flow will be diminished nearly in proportion to the square of the velocities, and we shall have $K^2V^2=9\times 25=225$. The value of r will also be somewhat diminished, and may be taken as 2, so that $r\frac{T_s-T_a}{3600}=0.123.$

$\therefore S=\frac{225\times 4.244\times 0.238\times 519.4}{64.4\times 50\times 0.123}=297.$ square feet.

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The heat expended per second will be $297 \times .123 = 36.53$ units per second, 2191.8 per minute, 131508. per hour.

$$T_c = \frac{36.53 + 1.01 \times 519.4}{1.01} = 555.5° = 96° \text{ F.}$$

$$D_c = 0.0808 \times \frac{491.4}{555.5} = .0719.$$

The theoretical velocity in the last case will be

$$V' = \sqrt{2g \times 50 \left(\frac{555. - 519.4}{519.4}\right)} = 15 \text{ feet per second;}$$

i.e., three times the actual velocity, which was assumed to be 5 feet.

The weight of air discharged will be $216000 \times 0.0719 =$ 15530 pounds. per hour, only slightly in excess of the quantity required, which was 15280 pounds.

It will be seen that the quantity K^n, the coefficient of the square of the velocity, and in regard to which the most uncertainty exists, has an important influence on the quantity S. It seems probable that a value of K between 3 and 4, or of K^n between 9 and 16, will answer for all ordinary cases, and that a value of r between 2 and 2.5 will be approximately near enough. The smaller values correspond to low velocities and large flues, and the larger values to high velocities and smaller flues.

If the value of V be assumed as 5 feet per second, and that of K as 3, or K^n as 9, then by combining the constants in equation (10) we may write approximately,

$$S = 1500. \frac{W T_a}{H(T_e - T_a)}$$

and for the area of the cross-section of the flue or flues for a given discharge (W) we have

$$A = \frac{W}{D_c V}$$

in which V, the velocity, is 5 feet per second.

These last equations are suggested as working formulæ under ordinary circumstances. Further experiments are desirable for ascertaining the values of K and r under ordinary conditions of the problem; still the formulæ, while only approximately correct, exhibit the laws which control the movements of the air.

To compare this method of ventilation with that by means

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of a fan or blower, let it be supposed, as found in the last example, that the heating surface required is, in round numbers, 300 square feet. The quantity of heat transferred per second would be 300×0.123 (nearly), or about 36.9 units of heat; per hour, 132840 units. If we estimate in round numbers about 1100 units to raise the temperature of water from 60° and evaporate it at 281°, corresponding to 50 pounds pressure, we shall have $\frac{132840}{1100} = 120.7$ pounds of water necessary to produce this amount of heat. Estimating one-horse power per hour at 40 pounds of water evaporated under the same conditions, the heat expended for ventilation would furnish in an engine about three-horse power.

Compared with the estimates of the horse power required for a Sturtevant blower to produce the same effect, the steam-coils are more economical than the blowers. There seems to be no doubt that steam-coils properly devised and adapted to chimneys or flues will give more efficient ventilation than the blower, for less cost of construction and maintenance.

A more extended discussion of this subject might be given, especially with reference to the constants in the equations; but the object of this paper is merely to open the discussion of the subject and to point out the correct method of treating the problem.

The arrangement of the steam-pipes in such a manner that the greatest amount of heat will be transferred to the air with the least resistance to its motion, is a matter of importance. It is desirable, if possible, to place these pipes within the flues, and it is a difficult matter to introduce the requisite amount of heating surface, if dependence is placed solely on the direct contact of the air with the steam-pipes.

The following plan is suggested for overcoming the difficulty. It has not, as far as I am aware, been heretofore practised. Vertical pipes, which are to contain live steam, are introduced into the flue, extending five, ten, or twelve feet upward in the flue. These are separated by sheet-iron diaphragms, which receive heat from the steam-pipes by direct radiation.

The flue is thus divided into several smaller flues at its base, the whole surface of these smaller flues, as well as the walls of the main flue, acting as heating surface. Air does not receive heat by radiation, but, on the other hand, it does not obstruct the radiation of heat.

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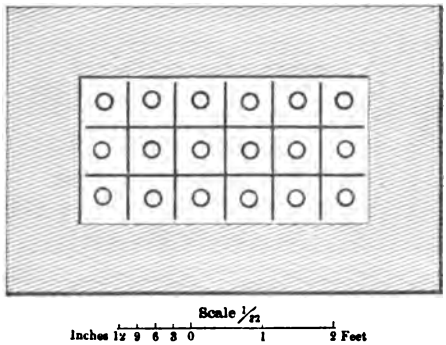
The figure below shows how a large amount of actual heating surface with a minimum amount of steam-pipe may be introduced into the base of a flue. In this case the flue has a cross-section of eight square feet, and it contains eighteen

CROSS-SECTION OF FLUE.



vertical steam-pipes, each two inches in diameter and twelve feet long, the pipes being closed at the top with suitable air-valves.

By means of the diaphragms shown, the actual heating surface with which the air comes in contact, is as follows: for each foot in height (of the 12 feet) of flue occupied by the pipes.

For the steam pipes, $\frac{5}{10} \times 18 = 9$ square feet.

For the walls of the diaphragms, including the flue walls, $2'.66 \times 18'. = 47'.88$ square feet.

Total per foot height..... 56.88 square feet.

Total for twelve feet high .....682.56 square feet.

Thus, for a total exterior surface of steam-pipes of 108 square feet, an actual efficient surface for heating the air is obtained of $682'.56$ square feet.

This surface may not all be as efficient as the surface of the pipes, but when it is considered that steam-pipes can give off heat by radiation without losing very sensibly their efficiency of heating air by contact under the conditions here given, it is evident that the surplus surface obtained by the diaphragms ought to compensate for the diminution of active pipe surface. Experiments alone will determine this question, but there seems to be no scientific reason why the device should not be effective.

In this case it will not be difficult to determine the co-efficient of velocity K by actual calculation from known data.

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ENDLESS-ROPE HAULAGE.

PLAN OF MECHANICAL TRANSPORTATION PROPOSED FOR THE BLOSSBURG MINE AT ARNOT, PENNA.

By W. BARCLAY PARSONS, Jr. ('82).

ABOUT ten years ago the North of England Mining Institute appointed a committee of engineers and coal-masters to fully investigate the subject of mechanical haulage of coal as at that time developed in the English collieries. After careful study the committee drew up an exhaustive and elaborate report, in which they classified the various systems into four groups, known as:

I. Tail-rope system.
II. Endless-chain system.
III. Endless-rope system No. 1.
IV. Endless-rope system No. 2.

It is not within the scope of the present article to discuss the differences and merits of the above systems further than simply an explanatory definition, and then to pass on to the description of what has been done in the way of mechanical haulage in the Blossburg district.

The Tail-Rope.—This system is the only one of the four that requires but one track. The rope is not continuous, but is broken and the two ends are fitted with hooks that are fastened to the front and rear of the train to be drawn in or out. The rope passes around a large drum at the engine end, which of necessity has to run in both directions according as trains are passing in or out. The peculiar excellence of this system is the ease of having stations and branch-lines worked by the one engine.

Endless Chain.—In the endless chain we have two tracks, one for the outgoing and the other for the ingoing trains. Motion is imparted to them by an endless chain running around a large wheel at the engine, which always turns in the same direction. The chain lies on the top of the cars, connection being made by a fork on each car into which the chain drops

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and catches. The cars are connected to the chain singly, and not in trains.

Endless Rope.—In both the endless-rope systems, wire rope, as the name implies, is used as the means of giving locomotion. In system No. 1, it, resting on rollers, passes under the trains that are connected to it by means of clamps or short chains. In No. 2 it passes over the cars like the endless chain, and is attached to them by short chains instead of forks.

Following the English definitions, as given above, we will proceed directly to the description of the endless-rope system No. 1 of the Blossburg district, Tioga Co., Pa., and especially the proposed plans for the mine of the Blossburg Coal Company at Arnot, as designed by S. B. Elliott, chief engineer of the company, and W. B. Parsons, Jr., assistant engineer. The principal collieries in this part of the country are located at Antrim, Fallbrook, Morris Run, and Arnot—the first three named being the property of the Fallbrook Coal Company, the latter of the Blossburg Company. The coal is semi-bituminous, showing, on analysis, carbon varying from sixty-two per cent. to seventy-five per cent. Of these four mines mechanical haulage has been introduced into two, Antrim and Morris Run, and proposed for Arnot, as hereafter described. At Antrim it did not prove a success, and, consequently, was abandoned some years ago, for the reason, the writer believes, of the many drifts leading into the mine, consequently the output, not being sufficiently concentrated in any one, did not warrant the expense. The coal is now drawn out by small locomotives.

In the mine at Morris Run, under the skilful management of Mr. Nearing, engineer and superintendent, the introduction of the rope has been attended with greater success. Although several drifts are in operation, yet here there is a main one, 2,270 feet in length, 16 feet wide, and perfectly straight, thus affording a splendid opportunity for rope haulage. The method adopted was, as previously stated, that of endless-rope system No. 1, where the rope passes under the trains. The rope is of steel, three-fourths of an inch in diameter, and of a single length of 2,280 feet. On coming from the mine it passes out on a trestle some 300 feet in length to the chutes. Power is supplied by two engines of 30-horse power each, which are run so as to give the rope a speed of about 500 feet per minute. The average grade is about 2½ feet in 100, while the steepest is 4½ feet in 100: this grade is "in-bye," so the

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heavy trains have to be pulled up it. The distance between the branches of the rope is 8 feet. The driving wheel, or the one around which the rope passes in order to receive motion, is 16 feet in diameter, composed of wooden arms and iron rim, and is very similar, as are many other details, to those of the projected plans for the Blossburg mine, about to be described. As regards the expense, it can be stated that the average cost of haulage by mules is 8½ cents per ton of coal, while with the endless rope, allowing for cost of rope 0.4 cent, for labor 2.32 cents, for machinery and plant 1 cent, we have a cost of 3.72 cents, thus showing a saving in favor of the endless rope of 4.78 cents per ton of coal.

The fourth and largest colliery of the previously mentioned list is that of the Blossburg Coal Company at Arnot, for which full and complete designs for haulage by the endless-rope system No. 1 were drawn up during the last summer.

This mine is the principal one of Tioga County, and on the property of the company there are two veins, the most important of which is known as the Bloss Vein, ranging in thickness from three to five feet. This vein has been worked through two drifts for about fifteen years, during which period a very large amount of coal has been removed. In fact, so far are the breasts from the mouth, in many cases nearly two miles, that the old method of hauling out the material in trains of five cars drawn by mules has become too laborious and expensive, and the necessity of resorting to mechanical means has become evident. Of the two drifts above referred to, known as Drift No. 2 and Drift No. 3, the former leads into a portion of the mine now almost worked out, while the latter serves as an entrance to a territory in which there is known to exist a large coal-field in addition to a considerable area still unexplored.

The coal, as was said before, is semi-bituminous, lying for the most part on a bed of fire-clay, though in places on sand-rock, and is covered by a slate of considerable strength. Through the coal run "cleats" or "joints," or, as they are termed by the miners, "slips," which have a bearing of 17° to the westward of north. So true do these cleats run that the principal drifts are driven entirely without the aid of instruments by following their direction. By running the headings in this way, the working chambers are cut at right angles to the slip, thus facilitating the operations of mining the coal. Drift No. 3, the one for which these plans are proposed, enters

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the side of the mountain with a bearing due north, and with a gradient just sufficient to give the water coming from the mine a motion outward, since the vein has a dip "out-bye." For the distance of 399.75 feet it keeps straight, and having encountered quicksands in its construction, it is heavily timbered and lagged on the sides and top. Owing to the expense there was left but seven feet clear between the side timbers, thus allowing room for only one track and the side ditch. At the end of 399.75 feet from the mouth the timberin stops, the drift turns to the left, and for the rest of its length follows the direction of the slip, which bears, as has been said, approximately N. 17° W. After running 850 feet the drift is widened to an average width of about 15.5 feet; two tracks are laid down for the full and empty trains, and the roof of slate is supported by a row of timbers set down the centre. The accompanying outline map, Fig. 1, explains more fully the foregoing and following details.

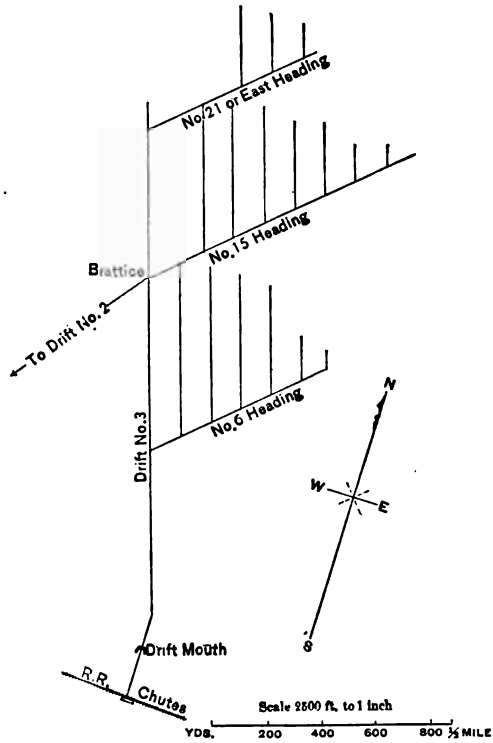

Fig. 1—OUTLINE MAP, DRIFT 3, BLOSSBURG MINE

From the main drift there are now in operation three diverging drifts which turn to the right to open up the coal-field in that direction, and are known in the miners' phraseology as No. 6, No. 15, and East Heading. They make an angle of 65° with the main drift, consequently their bearing is N. 48° E. The first of these, No. 6, turns away at a distance of 2,457 feet from the inner end of the timbers, or 2,857 feet from the drift's mouth. As the map shows, No. 6 is 2,600 feet in length before it turns, having nearly reached the prop-

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erty line, into a north heading, as the headings driven in the direction of the "slip" are termed. Between that point and where it leaves the main drift are five north headings. These headings are about 450 feet apart, and vary in length according to the time each has been worked. From the beginning of No. 6 to the point where No. 15 commences is about 2,200 feet. At the opening of No. 15 there is at present a heavy timber "brattice" across the main drift, for the purpose of regulating the proper current of air for ventilation. So the tracks, instead of continuing straight on, turn into 15 Heading. Thus the coal that passes through the main drift is that which is brought down through Headings 15 and 6 from their respective north headings. The length of 15 is 4,000 feet, the first heading north starting at 800 feet from the junction of 15 and the main drift, besides which there are seven other north headings. It must be understood that such figures as these, which are always on the increase, viz., the number of north headings, etc., are based upon a survey made in July, 1881, and of course are not strictly correct now, for between the time of surveying and September, No. 15 has been pushed on far enough to allow another north heading to be started. Continuing on with the main drift from the brattice at 15, a distance of 1,950 feet, we reach the East Heading. Practically this is a continuation, with a change of direction, of the main drift. Although it, the main drift, was driven straight on some 200 feet, the quality of the coal had deteriorated so much as to lead to the abandonment of pushing it any farther. This East Heading is 2,500 feet in length, with three North Headings, the first starting at a distance of 1,400 feet from the main drift. The coal from the East Heading is at present conveyed down Drift No. 3 to the brattice at 15, thence by a diagonal heading, somewhat over a mile in length, to the mouth of Drift No. 2. That part of this diagonal heading where it turns away from the main drift is shown in the map.

Such is Drift No. 3 underground. On the outside from its mouth there runs due south—that is to say, the continuation of its direction at the exit—a covered way 700 feet to the chutes, where the coal is dumped from the small mining cars into bins to be picked and cleaned, and thence loaded in large cars to be carried to the washer.

We now have before us the general features of the engineering problem to be solved. For the next ten years a

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very large amount of coal will be brought down from the headings described as East Heading, 15, and 6. This output can be estimated at a minimum of 800 tons per day. A rope even as far as 15 would have to be 6,000 feet in length, or 12,000 feet of rope, and as that is a long distance to operate, it immediately becomes a serious question whether there would be any economy or even possibility of continuing it on 2,000 feet farther to the East Heading, thus giving a total length of rope of 16,000 feet, or something over three miles. It certainly seems a far better plan to bring the East Heading coal down by mules to 15, and, having removed the brattice, to make the cars there up in trains and haul them out by the rope. This would have the additional advantage of obviating the necessity of an extra station at 15. In the question of which of the four systems to adopt, although the tail-rope would offer great facilities for side stations and the use of auxiliary ropes in Nos. 6 and 15, thus saving mules there, yet the slowness of a single track, and the delaying of loaded trains for the passing in of empties, outweighed the advantages. Of the endless-rope system No. 1, where the rope passes on rollers under the cars, and the endless chain and rope system No. 2, the former seemed the easiest to manage, and was accordingly adopted.

Having determined on that, it next became necessary to make an accurate and complete survey from the inner end of the timbering at the curve to the brattice at 15, in order to locate the sides of the drift with respect to a straight line connecting these points, to see whether any cutting of the walls would be required for the laying of a straight double track. A traverse was run through the drift, commencing with the inner end of the timbers and stopping just short of 15 Heading; from the data thus furnished a straight line was run, connecting a point in the last timber 4 feet 8 inches from left side wall with the centre point of the drift at 15 Heading. This line, 4,534 feet in length, was permanently located by stations in the roof, 200 feet apart, thus giving the centre line midway between the branches of the rope. At present the clear space between the side timbers is 7 feet 3 inches. To give plenty of room it would be necessary to take down the right or east side and set the timbers back so as to increase the distance to 9 feet 6 inches. The reason for this increase will be seen farther on. For the first 200 feet the line lay wholly in the right-hand wall,

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and then for the next 300 feet it is just clear of it, so for 500 feet the right-hand wall would have to be cut back several feet. After that the drift is now much wider, and only cutting here and there in a few places would be necessary in order to give a minimum distance of 5 feet 9 inches on either side of the survey line. With these alterations the drift would be in shape to lay a straight double track 2 feet 2 inches gauge with 8 feet between track centres except through the timbering, where it would be contracted to 4 feet 6 inches. This is in order to allow sufficient clearance, since the drift is here narrowed to 9 feet 6 inches. With cars 3 feet 6 inches wide this gives 1 foot between them and 9 inches clear between their sides and the timbers. This narrowed condition lasts for about 800 feet.

On taking the levels of these survey stations the total rise of the drift between the timbers and No. 15 was found to be 82.65 feet, giving an average of 1 in 54.8. The sharpest incline is 74 feet previous to the termination of the survey line, where for 60 feet it is 1 foot in 9.6. The slightest is through the timbers, where there is only sufficient grade to give water a good flow. But as the vein is worked upward the dip is "out-bye," the reverse of the Morris Run Mine, thus acting in favor of the heavy trains.

By referring once more to Fig. 1 it will be seen that from the drift mouth to the chutes is a distance of 700 feet, which is, as has been said, a covered way. For half its length it lies on the natural surface of the ground, while after that it rests on an embankment of "bony," or coal unfit to burn, having 33 per cent. ash. From the drift mouth to the railroad is 650 feet, where the height of the bony fill is 23 feet, and it is here proposed to place the engine with the necessary driving-wheels and guide-pulleys. The bony is to be excavated to give sufficient room, while the wagon-way and wheels are supported by suitable framework. The engine is a single-acting horizontal one, with a stroke of 18″. The cylinder originally had an internal diameter of 12 inches, but has since been bored to 12½ inches. Taking an average steam-pressure of 60 lbs., with 120 revolutions per minute, we deduce 25 as a value of its horse-power. The boiler is of the tubular type, being that of a 36-ton 6-foot gauge locomotive. To prevent the chance of fire being communicated to the bony, the boiler will be placed in a separate building at a dis-

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tance from the engine, while the steam is conducted underground in carefully covered pipes.

The shaft, which is 6 inches in diameter, is to be placed with its length north and south, or parallel to the wagon-way, 16 feet above it. Hence, the rope to pass around it has to bend from its horizontal position to the vertical. The wheel, shown in Plate 1, around which it turns, is of considerable importance. In order to give a greater bearing-surface, to increase the friction, and also to prevent too rapid wear of the rope by a sharp bend, the effective diameter is taken at 16 feet, *i.e.*, the distance from centre to centre of the rope when in position. As is seen from the drawings, there are four parts, the rim, hub, tie-rods, and arms. The first, shown in section in Fig. 4, is cast in halves, the front and rear, each half being again subdivided into pieces forming arcs, subtending angles of 30° at the centre. The two sides are held together by $\frac{5}{8}$ inch bolts, in such a manner that the separate pieces break joints with each other. The rim is strengthened by a reinforcement on each side. In Fig. 4 it is shown that the rope rests on wood, as does the rope in a very similar wheel at Morris Run; but it is quite likely that hard rubber would be substituted in its place. It is usual in wheels of this description to make the guide-arms that keep the rope in place of considerable length; but in this case, inasmuch as the wheel will not revolve at a high rate of speed, and the rope is always kept in a state of tension, a length of $1\frac{3}{4}$ inch from the centre of the rope was considered ample, thus giving an extreme diameter of 16 feet $3\frac{1}{2}$ inches. In order to give the rim support at as many points as possible without increasing the weight, wooden arms were adopted instead of iron, and since the principal strain is one of compression, they answer very well. They have a thickness of 3 inches, a width at the top of 4 inches, and at the exterior of the hub of 5 inches, and are twenty-four in number, being set at an angle of 15° to each other. To the rim they are fastened as shown in Fig. 4, being held by a $\frac{5}{8}$-inch bolt to the projecting tongue. A rough calculation puts about 1,900 pounds of metal in the rim. The hub, shown in section in Fig. 3, has a length on the shaft of 15 inches. The portion to which are attached the arms is a central disc $\frac{3}{4}$ inch thick, with a diameter of 3 feet 4 inches. Serving the double purpose of braces to the disc and partitions between the wooden arms are twenty-four ribs, each 3

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inches thick and 1 inch wide, whose outer ends are on the circumference of a circle with a diameter of 3 feet 10 inches. The arms, having a mortise cut into them, are slipped over this disc between the ribs. To withstand any strain arising from centrifugal force the twelve $\frac{3}{4}$-inch tie-rods are introduced. They are hooked into projecting eyes on the rim, while their other ends, passing through lips on the side of the hub (Fig 3), are secured by nuts. This is a simpler method of tightening than by turnbuckles. To keep the strain symmetrically distributed, the rods alternate on the sides of the hub. With a factor of safety of 3, each rod is capable of bearing $3\frac{1}{2}$ tons. The engine is connected with the shaft by gearing in such a way that, while the engine makes 120 revolutions, the shaft makes but 14, thus giving the rope a speed of 8 miles an hour.

As the driving-wheel has a diameter of 16 feet, and the distance between the branches of the rope on the wagon-way is

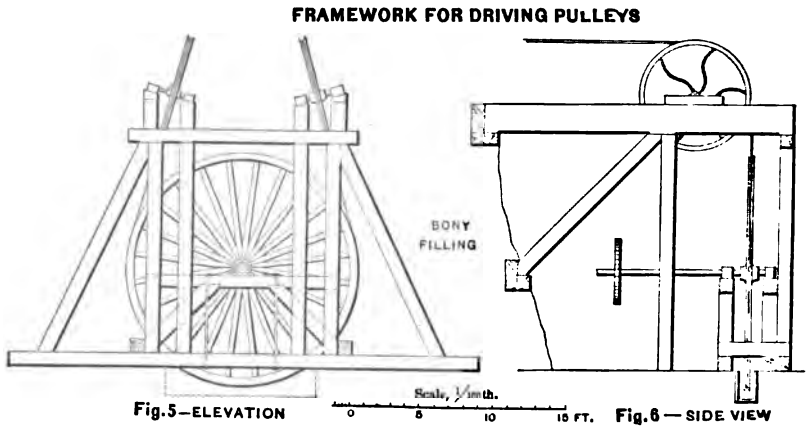

Fig.5—ELEVATION Fig.6—SIDE VIEW

8 feet, it is evident that the rope must wrap more than 180°. Letting it encircle 210°—that is, have the branches make an angle of 15° with the vertical—it is necessary to place the shaft 16 feet below the plane where the horizontal distance between them is 8 feet. The guide-pulleys to effect the change of direction are the ordinary wheels constructed for the purpose by Messrs. Roebling & Sons. They have a diameter of 8 feet, and of course must be inclined 15°. The timber-work for their support and that of the large driving-wheel is shown in Figs. 5 and 6, end and side elevations. As the shaft is to be 16 feet below the wagon-way, and the extreme radius of the wheel

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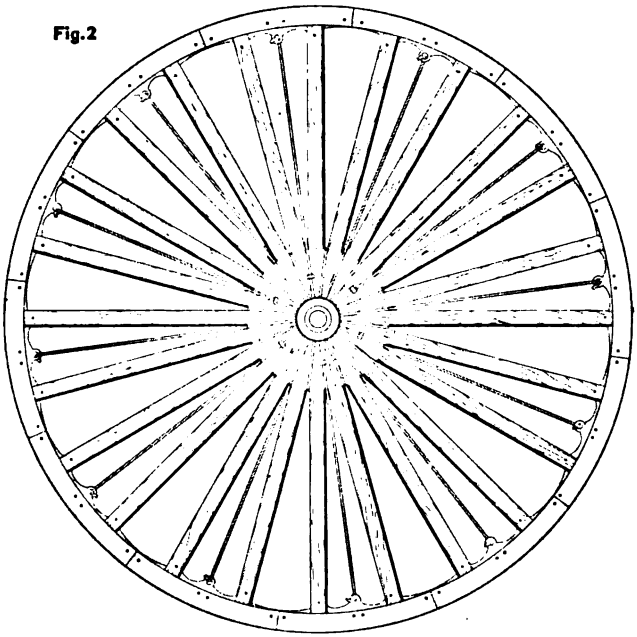
DRIVING WHEEL

Fig.2

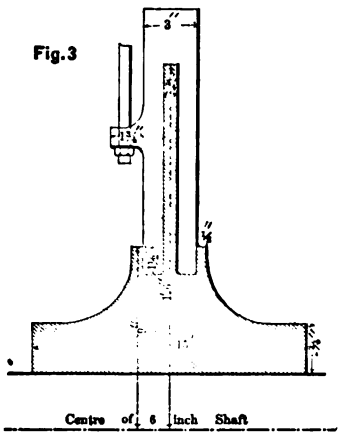


ELEVATION

Scale, 1/40 th.

0 1 2 3 4 5 FEET

Fig.3



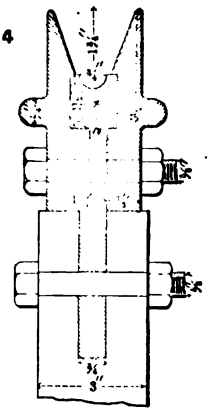
HUB

Scale 3/10 th.

8 9 12 IN.

CROSS SECTIONS

Fig.4



RIM

Scale 1/5 th.

2 6 IN.

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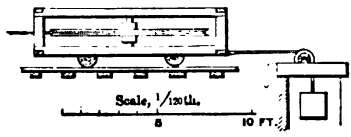
being 8 feet 1¾ inches, the bottom of the wheel will be over 24 feet below the wagon-way. But the height of the bony fill is only 23 feet, hence the wheel will be slightly below the ground. To accommodate it, an excavation 2 feet deep, 1 foot wide, and 10 feet 6 inches long is to be dug, the sides of which are to be protected with 2-inch plank; 6 inches from each side of this trench are laid the foundation-sills for the support of the driving-wheel. On the outside lies a long sill for the framing above. The three are held together by stout cross-ties. All the pieces, with the exception of the long stringers, which are 12 × 24 inches, have a cross-section of 12 × 12 inches. The construction of the framework is best understood from an inspection of the illustrations. The ends of the long stringers and diagonals, whose length is in the direction of the thrust of the resultant strain on the guide-pulleys, which are shown in the side elevation, rest on timbers firmly embedded in the bony. The tops of the stringers where the journals of the guide-pulleys rest are bevelled off at an angle of 15°, to give the wheels the required inclination. Where timbers meet they are to be secured to each other by mortise-and-tenon joints and stout dowels or pins. Through the string-pieces, on which the journals of the driving-wheel rest, the diagonals, and the lower sills, are inserted 1-inch tie-rods, of which the heads and nuts press on plates of metal, to give a large bearing surface on the wood. They are to hold the pieces together more securely, and also to resist any upward motion of the string-piece arising from the tension of the rope on the wheel.

Should it be decided to run the rope only as far as 15 heading, the turning-point of it will be placed about 100 feet beyond the termination of the survey line, or 5,034 feet from the drift-mouth, giving a total single length of rope of 5,700 feet. The turning of the rope will be around a horizontal wheel 8 feet in diameter, and identical with the guide-pulleys at the other end of the line; it will be held in a stout timber framework carriage, as shown in Fig. 7. These wheels are furnished by Messrs. Roebling & Sons at a cost of $125. In order to take in the slack arising from stretching, and to keep the rope at an even degree of tension, there are two methods: one to connect this carriage to a post by means of turnbuckles, the other to keep

Fig. 7—TIGHTENING WHEEL



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it in place by a counterpoise of sufficient weight, as can be understood from the same drawing. This method is by all means the simplest and the most convenient. A pit is dug in which hangs the weight.

Messrs. Roebling & Sons estimated that a steel rope ¾ inch in diameter would be sufficiently strong to bear the strain. Such rope would have a breaking strain of 15 tons and a working strain of 5,000 pounds. The rope would rest on wooden rollers about 8 inches in diameter, set 50 feet apart.

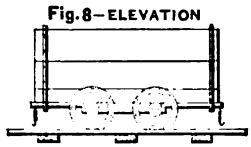
Fig. 8—ELEVATION

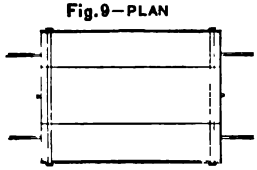
Fig. 9—PLAN

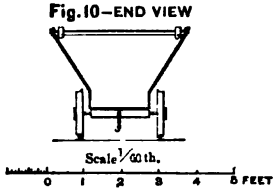

Fig. 10—END VIEW

We have now described the permanent way, with details of machinery, etc., and proceed to that of the cars and the manner of running the rope. The cars or tubs used in the mine are like the ordinary small mining-car—Figs. 8, 9, 10. They are adapted for a gauge of 2 feet 2 inches. Unloaded, they weigh 550 pounds, and can carry a burden of 1,200 pounds. They are fitted with the Davis patent wheel. Supposing the rope not to be carried farther than 15 heading, it will be seen, on referring to the map, that there will then be two stations or places where trains will have to be attached, viz., 15 and 6 headings. In order to have the cars cross the rope, the simple arrangement of the switch shown in Fig. 11 is devised. The rail *A* is short and movable, being hinged at *B*. When the train is to pass along the main line, it is left in position as shown; when, however, the train is to pass on to the switch, it lies as indicated by the dotted line.

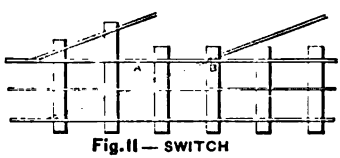
Fig. 11—SWITCH

To give an account of the method of manipulating, we will suppose we have a loaded train at 15 Heading, of the full complement of cars, 30. They are linked together in the ordinary manner, and at each end of the train is attached a small car just large enough for a man to sit on. These are the men that have control of the train, and are called run-riders. It is their

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especial duty to couple the trains to the rope by means of a clamp. The clamp or clutch used in England is similar to a pair of tongs, quite large and clumsy. The one used at Morris has two circular wedges as the tightening power. In most mines there is need of but one connecting clamp, in front of the train, if the drift dips inward and the train has to be pulled out, or at the rear of the train in the reverse case. In the present instance, however, though the dip is outward, and at the beginning at 15 Heading is sufficient to run the trains at a rate considerably faster than 8 miles an hour, yet at the mouth it is not sufficient to overcome friction. On this account two clamps are necessary.

To resume the explanation of managing trains: before the train can be attached the rope is stopped, the cars then being run out from the heading on to the main line. When the rear run-rider's car has passed over the rope, the clamp is attached, and the brakes that kept the cars from running down hill being released, the train by its own weight stretches itself out to its whole length; then the front clamp is attached. All being in readiness, a signal by appropriate means is made to the engineer in charge of the engine, who sets the rope in motion. In the meantime, at the engine end a train of empties has been made up and attached by a single clamp, to be drawn in as the full train comes out. After coming down the last steep hill in the mine, the rear clamp is disengaged to give the foremost run-rider complete control. Just before the train reaches the guide-pulleys, the wagon-way is slightly raised so as to allow the train, on the clamp being unfastened, to pass away from the rope, and to run by the momentum imparted to it 50 feet farther on to the chutes. In order to prevent an accident in case of failure to disengage the clamp, a block, connected by a wire with a brake on the gearing, is placed by the rope, against which the clamp would strike, and thus automatically stop the engine.

In England the method of sending signals from the inside to the engineer is by means of what is termed there a sounding-rod, which is simply a rod of iron, which, being struck, conducts the sound to the other end. But, on account of the length in this case, the use of electric bells is suggested. A copper wire could be laid along the roof, and at Headings 15 and 6 the small push-buttons could be attached to ring a bell in the engine-room. Besides these necessary ones, there might

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be supplementary ones every 300 to 400 feet, to ring to stop the rope in case of accident to a train while in transit. Since leaving Arnot the writer has learned of a successful adoption of an electric system of signals at the Hazard collieries, near Liege, Belgium.

To give any calculations as regards strain on the rope, the amount of work to be done by the engine, etc., would be purely theoretical, and practically not to be depended on, for the lack of fundamental data such as coefficient of friction, etc., for these particular circumstances. In England, at the colleries of the Wigan Coal and Iron Company, a carefully conducted series of experiments were made by the committee of the Mining Institute, to determine the coefficient of friction for mine-cars. They used two methods of investigation, giving as results 0.2092 and 0.2548. But as the car-wheels at Arnot are of an improved pattern, even the lowest of these determinations must be considered too high. The fact, too, of the grade sloping outward will relieve the engine of a very great part of the strain, as the full cars running out will help, and in some places entirely suffice, to draw the empties in ; hence, the work will not be so heavy as one might at first suppose would arise from an endless wire rope with a single length of 5,700 feet. Of curves in the line there will be three : one to narrow the distance between the branches of the rope on reaching the drift-mouth, one to widen it again at the proper place in the mine, and a third at 400 feet inside from the drift-mouth, where, as was said before, the drift turns to the left 17°. All these will be so slight as to present no serious obstacle, as in several cases in England endless-rope railways have curves of a far shorter radius than will be necessary here. The rope on curves will be in large wooden sheaves, to pass around which English experience has shown the clamp, since it is not rigidly fastened to the train, finds no difficulty. A wire rope lasts about three years.

As regards the amount of work expected from such a system of haulage, it is calculated that the output through the drift from Headings 15 and 6 would be 800 tons a day, about 600 to 700 tons being the present output as drawn in trains of five cars by mules. To get the amount of 800 tons it is calculated to run about five trains an hour for a working day of ten hours, and thirty cars to a train. The average capacity of a car has been given before to be from 1,000 to 1,200 pounds.

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The speed of the rope would be 8 miles an hour. In England the highest speed is at the California Pit, near Wigan, and is 11.7 miles an hour on tangents and 7 miles on curves. This, however, is on a much shorter line with sharper curves; so at Arnot the trains can be kept up to the speed of 8 miles an hour through the entire length of the line.

These designs were intended for, as was said in the beginning, the mine of the Blossburg Coal Company; but, a change in the direction of the company taking place, work on them has never been anything further than the drawing up of the foregoing plans and specifications, and the making of the survey as described. Whether operations will be resumed at a future date is at present undecided. That some system of mechanical haulage in a mine of such dimensions and so great an output is required is unquestionable, as every day of the present method of draft by mules is a direct and decided loss, as is shown by the figures previously given of the saving at Morris Run. Moreover, the coal all coming down from two, or, at most, three headings, in which latter case the output would be about 1,200 tons instead of 800 tons per day, offers an unexceptionable opportunity and facility for the making up and sending out of large trains, much more so than if the drift was tapped at several points. As regards the economy of mechanical haulage, there is a rule in England that it pays as soon as the number of horses required exceeds six; but in this country the superiority of our mules and the less expense of keeping them would somewhat raise that limit. In respect to the expense of such a plant, it would be very difficult to give any figures in advance of the work, as really no estimation of how much is to be done in retimbering the drift-mouth can be made beforehand on account of the presence of quicksands

In conclusion, the writer wishes to acknowledge his indebtedness to Mr. E. G. Tuttle, of the class of '80, S. of M., for much of the data relating to the mine at Morris Run; but more especially to express his thanks to S. B. Elliott, Esq., Chief Engineer of the Blossburg Coal Company and Tioga Railroad, for kindly allowing him to copy the designs of the proposed work, and also to publish the information contained in this article.

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THE MARQUETTE IRON REGION.—III.

By THE STUDENTS[1] OF THE SUMMER SCHOOL OF PRACTICAL MINING, LAKE SUPERIOR, 1881.

DRILLING AND BLASTING.

MACHINE-DRILLS are in very general use in the mines. In the last number of the QUARTERLY a table was given showing the comparative speeds of hand and machine work in shaft-sinking at the Barnum Mine.

At the Republic Mine machine-drills are used wherever it is possible to set them up. Rand, Ingersoll, and Waring machines are used, the former being the favorite with the miners, although its tripod is disliked because only one leg is adjustable. The Ingersoll has two legs adjustable, and the Waring all three. All of the tripods have the same arrangement for swinging the drill. The bits are in lengths of 2, 4, 6, and 8 feet, and are of two styles—burr and single edge, the width on the edge being from 1 inch to 2¼ inches. The men are paid about twenty-five cents per foot of hole drilled.

Ingersoll and Rand drills are employed at the Champion Mine, the tripod of the former being used in sinking and stoping, and the column-support in drifting and cross-cutting.

At the Michigamme Mine the drilling is nearly all done by machine-drills, of which the company have thirteen, seven Rand and six National. In stoping, large-sized holes are bored up to 2¼ inches in diameter, and heavy charges are used, the holes sometimes being 12 and 15 feet deep. In drifting and sinking, the maximum diameter of holes is 1½ inch, and 8 feet the greatest depth.

All of the drilling in the Jackson, and nearly all in the Lake Superior Hematite Mines, is done by hand, one Ingersoll drill and one Waring drill being used at the latter mine. Much hand-work is also done at the Cleveland. Machine-drills are used in the other mines visited as follows: New York—Rand, Burleigh, Ingersoll; Lake Superior—Rand.

[1] Being extracts from the memoirs required of each student of the Summer School.

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Explosives.—At the Republic Mine both nitro-glycerine and giant-powder are used, the latter especially in places where a very violent shock would be apt to bring down the sides and roof. The quantity used depends upon the judgment of the miner, no particular rule being followed. Nitro-glycerine is put into tin cartridges closed with a cork, through a notch in which the fuse passes to a copper cap containing some detonating material, as fulminate of mercury. Double-taped fuse is exclusively employed. Ordinarily no tamping is used with nitro-glycerine, but when the blast-hole is in the roof, *i.e.*, "looks up," wet clay is placed at the bottom and in the mouth to keep the cartridge in position. The charges are reckoned by the pound, and one man is detailed to do the charging and firing.

The Champion Mine uses both giant-powder and nitro-glycerine, limiting the use of the latter, however, to stoping. Holes 10 or 12 feet deep are bored, filled with water, and the "oil" then being poured in, settles gradually to the bottom by reason of its greater specific gravity. The water at first breaks the fall of the oil and afterward serves as tamping. In drifting and sinking the holes are charged with oiled-paper cartridges of giant-powder and tamped with 18 inches of dried sludge, carefully rammed with a wooden stick. These cartridges are fired by a high-tension electric battery. Seldom more than eight to twelve holes are fired in the same drift at once, and for this number the wires are arranged in series. The advantages of electric firing are: 1st, greater safety for the miners; 2d, economy of explosives; and 3d, economy of time; and these advantages are especially noticeable in shaft-sinking and drifting, where electric firing becomes most advantageous.

In the sinking of Shaft A, in the Barnum Mine, fourteen to eighteen holes, charged with No. 2 dynamite, are fired simultaneously by electricity from a low-tension machine. The composition of the dynamite is approximately 46 per cent. nitro-glycerine and 54 per cent. fine black powder.

At the Lake Superior Mine, giant-powder No. 3 and ordinary black powder are employed, the latter being most applicable to soft ore on account of its slower action. In using black powder at the Lake Superior Hematite Mine the rock is first "seamed" by firing 1 to 3 inches of a No. 2 dynamite cartridge at the bottom of each drill-hole, after which common black powder is poured in through gas-pipe and fired. The

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giant-powder used at this mine in the harder specular ore is manufactured by the Atlantic Giant-Powder Co. of New York, and contains 30 to 40 per cent. nitro-glycerine, and 60 to 70 per cent. of dope, consisting of nitrate of soda mixed with a small proportion of sawdust. It is put up in paper cartridges or sticks 1½ inch in diameter by 8 inches long, and is fired by the Bickford safety-fuse, with a detonating cap of German manufacture clinched to its lower end. The tamping consists of the mine dirt from the foot of the stopes, and the general rule followed is to use two-thirds by volume of tamping to one-third of powder.

Glukodine is exclusively used at the Cleveland Mine, the charges varying from an ounce for block-holing, to five or six pounds for large blasts. The tamping is fine ore or rock, from the floor of the workings. Glukodine was used for a while at the Michigamme Mine, but was abandoned after two accidental explosions, which resulted in serious injuries to several men.

Giant, Hercules, and common black powders are used at the Jackson Mine, the only rule in regard to the use of explosives being, *use black powder whenever possible*. The charges are fired by electricity, either singly or in roughly arranged groups, no attempt be made to secure simultaneous or rapidly successive discharges.

[It will be seen, from the above notes, that the use of high explosives is almost universal in the iron region. The tendency among the miners has been to demand more and more powerful explosives, the 30 per cent. dynamites first introduced being now replaced in the market by stronger powders, containing 40 per cent. and 60 per cent. of nitro-glycerine which have been manufactured to meet this demand. At some of the mines, in spite of the danger attending its use, nitroglycerine itself is employed.

There are, however, signs of a reaction in favor of less powerful and cheaper explosives. It is stated, on good authority, that, notwithstanding the introduction and use of explosives of increased strength and greater cost, the amount of explosive consumed per ton of ore mined is, throughout the region, as great as, and, in isolated cases, even greater than when black powder alone was employed. It is evident that the miners have not, as yet, acquired the skill in the economical and effective use of the high explosives that they had reached in the employment of black powder. In fact, the very strength

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of the powder is too often made an excuse for not exercising skill and judgment in the placing and pointing of the blast-holes. A part of this is, undoubtedly, due to the general use of machine-drills, with joints less flexible than those of the human frame, and therefore not always capable of boring holes in the best place and in the most advantageous direction. Another source of increased consumption is in the unnecessary diameter of the drill-holes, both machine- and hand-bored, which hold more powder than required for the work, being proportioned for black powder, but which the miners think it necessary to fill to the same extent as if they were using black powder. On the other hand, such holes, filled with high explosive, undoubtedly do more execution than if filled with black powder, breaking more ground, and thus lessening the labor of drilling per ton of ore mined, and also breaking the mineral into smaller pieces, and thus saving in labor of sledging and block-holing. The extra cost of the explosive is thus, in part or wholly, balanced by saving in cost of labor. There is, however, no reason why, with a more intelligent and economical use of the powerful explosive, these advantages need be lost. The subject is one which will repay study and investigation.—ED.]

Scramming.—A scrammer is a miner who prefers to work for himself, and takes a small contract, to mine ore from the sides of old pits, or from narrow portions of abandoned or worked-out deposits. The scrammer is paid so much a ton for the ore produced, he paying all expenses, for hoisting, powder, etc. Sometimes the scrammer takes a contract to work over the piles of waste rock, breaking up the large pieces and picking the good ore from the jasper.

HANDLING THE ORE AND WASTE ROCK UNDERGROUND.

Jackson Mine.—The ore when blasted down is loaded at once into cars, if not in pieces over ten inches in diameter; if larger, it is broken on the spot, by sledging or block-holing, to the proper size. If ore and rock are in the same piece and it is large, it is broken or blasted to separate them, and the selection is made on the spot. In the pits where the scrammers are taking down ore from the sides, and no sinking is in progress, buckets or cars are loaded by shovelling, and the rock is left at the bottom; but in the stopes all that is blasted rolls

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down the incline and is separated at the tracks. In most cases in the stopes the rock is also of necessity carried out, but it is only hoisted a short distance and then dumped into special chutes, which carry it away to deserted parts of the mine. In the stopes the loading is done directly from the protecting battery, at the bottom, into the cars; but, in the open pits, barrows are necessary, and in several places in these open pits, where a track would be inconvenient to reach, a bucket, worked by a large derrick, is used, and by its great swing saves much wheeling of ore at the bottom. These buckets hold about one ton, and their hoisting-ropes go to the engine-house drums in the same way as the skip-ropes.

Block-holing is thus conducted in the Cleveland Mine. For pieces weighing one ton or more the block-holes vary from 8 to 12 inches in depth, are charged with 1 inch to 1½ inches of glukodine, and lightly tamped with fine ore. To save time of the workmen a large number of holes are usually fired at once.

The men who load the underground cars are usually paid by the skip-load, and have, therefore, very little interest in sending up pure ore; hence, a man or boy is employed to throw out any rock or poor ore which may have been thrown in.

Republic Mine.—The process of getting the ore to suitable size for hoisting is much the same as that already given. If the ore shows a clear grain, it can generally be split with a "gad" or short, blunt chisel.

The cars used are mounted on four wheels placed near the centre, have sides of iron, and swinging ends of wood braced by iron cross-pieces which are held shut by latches at each side near the bottom. They are run directly up on the cages and hoisted to the surface without unloading. Their weight is 1,800 pounds, capacity about one ton, and they are moved by men or mules, the latter being sent down in the morning and raised at night in stalls on the cages.

The track is 3 feet guage, and 28 pound rails are used. The grade is as high as 1½ in 100, and the sharpest curve has a radius of 14 feet, but the arrangement of the car-wheels near the centre would permit even sharper curves than this to be used.

Lake Superior Mine.—Twenty-pound steel sledges are employed for breaking the ore, which is then sorted, the rock being piled on one side, while the ore itself is thrown directly into a tram-car.

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Larger pieces are broken by block-holing, the holes for which, varying from 8 inches to 16 inches in depth, are hand-drilled by a 1¾-inch bit struck by a 7½-pound hammer.

The ore is sorted almost entirely underground, an experienced miner readily distinguishing the different grades both by the weight and the peculiar ring, which in first-class ore is dull and heavy, while that of a poorer quality has a characteristic "chainy" resonance when struck.

New York Mine.—As the ore is blasted down in the stopes, it is broken into pieces that can be lifted by one man and loaded into barrows, from which it is dumped into the skips from a loading platform or plank laid across the shaft. The men who do this work are called fillers and are paid by the amount of ore sent up. The breaking-up is accomplished either with sledges or by block-holing, the latter being generally done with nitro-glycerine.

In some of the pits light enough enters to allow of sorting the ore underground, and the rock and ore are sent up separately. In other cases each skip-load contains more or less rock, and must be picked at the surface.

At the *Michigamme Mine* the number of trammers is about equal to the number of miners. It is the business of the trammers to sort and tram the rock and ore from the stope to the skip. The sorting underground is quite simple, three classes only being made, "rock" or waste, "Persberg" or second-class ore, and "Standard," or first quality ore containing 65 per cent. metallic iron and ore. The ore breaks off quite clean from the south or hanging wall, and here the broken ground consists of "rock" and standard ore only, no second-class ore being made. The rock is thrown to one side till enough accumulates to make a skip-load, when it is loaded and sent to the surface, a special signal being given to announce the character of the load. The second quality, or "Persberg" ore, occurs near the foot-wall. This is separated by the trammers in loading. The jasper and the mixed ore of the foot-wall itself is sent up as rock, but as far as possible the blasting is so managed as to break ore only.

The cars are loaded by hand, the lumps being inspected in the loading. Fine ore is not sorted underground, but is shovelled into the car and picked over on the surface.

Pieces too large to be lifted are broken by a sledge of 18 to 24 pounds—one man of a party of four trammers being en-

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gaged at this work most of the time. If the pieces are too large to be sledged, they are broken by "block-holing," that is, blasted by means of a small hole drilled by hand. Block-holing is the work of the stopers, but when the trammers are short of ore the work is done by them, and charged against the stopers at the rate of fifty to eighty-five cents per foot of hole drilled.

The tramways are so laid that there shall be a slight grade in favor of the load from the stope to the loading-plat at the shaft. Heavy rails are used and are laid without ties on the solid ore, each rail being laid in narrow channels cut with the pick or moil, and held in place by dirt tamped about them. This permits the ore to be blasted down on the track without danger of bending or dislodging the rails. The tram-cars hold from two to two and one-half tons, and dump directly into the skip, which stops just below the plat. To prevent ore and rock from falling between the skip and the plat, and down to the lower levels, movable boards are used, forming a sort of chute or hopper, which boards are turned back during hoisting and lowering of the skip.

HANDLING AT THE SURFACE.

In some few cases the ore passes from the skip directly to the pocket, and to the ore-car, as at the Jackson, where most of the mine being in direct communication with the railroad, the ore is hoisted just a little above that level and dumped into pockets which connect by chutes with the cars. In some parts of the mine the railroad is too distant, and then the ore falls through chutes into wagons and carts, in which it is hauled to the loading platform by the track. The buckets, when used, dump their contents into carts which drive up to the edge of the pit. The same general methods are employed at the Cleveland Mine, where all the shaft-houses are either directly over a branch railway, or have a chute to convey the ore to the cars, so that the skip empties its load at once into the cars or into the pocket above them. Each skip-road, with the exception of that at the Saw-Mill pit, where the ore is not sorted, is provided with a pocket. At other mines, however, the ore has generally to be trammed from the shaft-houses to the pockets in hand-cars, or in cars drawn by horses or mules.

The shaft-houses of the Champion Mine are built directly over the mouth of the shaft, the skip-way extends to the top of the shaft-house, and behind it is a wedge-shaped bin ex-

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tending from about 5′ above the floor to one-half the height of the building. This bin is divided into two compartments by a partition through the middle; directly over the partition, and under the self-dumping device in the skip roadway, is the inlet into the two bins, the remaining portion being boarded over. Hung to the partition in this inlet is a door, which can be swung backward and forward, thus closing at any time the entrance into either bin. By this contrivance the good ore and the rock are made to dump into separate bins. On the outside of the shaft-house are chutes, closed by cast-iron aprons, which can be raised and lowered by long levers and a windlass. The ore falls through these chutes to a car, and is hauled to the ore-pockets or stock-piles by horses.

At the New York Mine the skip-roads from the different pits extend far enough above the surface to allow the dumping of the skips into cars running on trestle-work. These cars are run along the trestle-work and dumped on the stock-piles, or into chutes or short slides, leading to the railroad cars below. In some cases the railroad track runs near enough for the ore to be dumped upon the ore-cars directly from the skip. To facilitate examination and separation of second-class ore and rock, the ore hoisted during the day-shift is dumped on a short slide, as above mentioned, a portion falling directly into the car and part remaining on the slide. All dirt and dust is washed from the ore, by water from a hose, so that good ore may be readily distinguished, and all doubtful pieces are thrown aside as waste, the remainder being scraped into the cars. The different pits being let on contract, each pit has a sorter at the surface, who helps pick over the ore and records the numbers of the cars that are loaded with ore from his pit. The cars are weighed at Escanaba, and each contractor credited with the weights corresponding to these car numbers.

The ore from Pit 5, being largely mixed with jasper, is not dumped directly into the cars, but in a pile on the ground, when the good ore is picked out, loaded into a cart, and drawn about 100 feet to a platform, from which it is dumped into the car below. The ore hoisted during the night also is not dumped in the cars, but on the stock-pile, record being kept of the amount, and the contractor credited with a certain weight per skip. When the ore is not uniform, each skip-load is weighed and a certain weight deducted for waste.

At shafts 1, 2, and 3, of the Republic Mine, much the

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same general methods are followed, but at shaft No. 6 the arrangements are far more complicated. At one side are three burrows, or stock-piles: one for black ore (magnetite), one for slate ore (specular), and one for second-class ore. The second class and magnetite are conveyed to pockets on the main track, in front of No. 3, by a line of tram-road, and the slate-ore is carried to pockets by six parallel lines of tramway. The slate or specular ore, as it comes from the mine, is either dumped on the stock-pile or run down a self-acting incline, 270 feet long with slope of 1 in 10, to pockets over the railroad. The rails over each pocket are provided with hinged segments, which, when raised, expose slight depressions, into which the front wheels of the car run, by which the cars are dumped. The trammers sometimes have to assist this operation with the aid of a pole inserted between the wheels, with which they tip the car and allow the contents to fall to the pocket below.

Michigamme Mine.—The arrangements for handling ore on the surface are very complete, the ore being lifted by the shovel or by hand but once from the time it is blasted down till it reaches the deck or hold of the vessel at the shipping docks at L'Anse, excepting only that ore which has to be stored at the mine on the stock-piles.

The signals announcing the nature of the up-coming skip-load, the switches are set for tramming the load to its proper place. If the load is rock it passes directly to the dumps, most of which are so arranged as to fill up old workings. If the load be Persberg, or second-class ore, it goes to the stock-piles; or, if standard ore, it goes to the picking-table over the pockets. In every case the tramway has a slight grade in favor of the loaded cars. From shaft No. 2 there is a gravity road to the pockets. From No. 4 the cars are drawn by a mule. From the open pits the cars are smaller and the tramming is by man-power.

There are nine pockets, each holding about 150 tons. They are really underground, being constructed in the old surface workings. The railroad cars run under the pockets by sidings from the main line. These cars hold 6 to 7 tons, and at present (July 23d) 81 cars are loaded per day—65 going to L'Anse and 16 to Escanaba.

Persberg ore, for which there has been but little sale until lately, is loaded from the stock-piles by barrows and movable trestles.

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The appliances and method of picking the standard ore deserve special mention. The ore as it comes from the mine is moist, often so coated with fine rock and ore that its grade cannot always be determined with certainty. Hinged platforms are therefore provided, on which the car-load of ore is dumped and washed with a hose. The pickers then examine the ore and throw out any waste or second-class ore. After this inspection the button is turned and the ore slides into the pocket below. By this simple device, the invention of Captain Christopher, two men are able to pick 90 tons of ore per shift, at a cost of $3\frac{5}{8}$ cents per ton for labor.

Pockets.—Pockets are made of heavy timber, sometimes lined on the bottom and sides with T-rails. Those which are placed at the side of the railroad track slope toward it, and are closed by gates worked by hand-levers and balancing weights, or by small winches placed above.

Those which are over the track have the sides sloping to form a kind of hopper, and are supported on a heavy framework, which allows the cars to pass underneath. The small lower end is closed by drawing-bars.

Stock-Piles.—During the winter no shipments are made, and the ore is piled up at a short distance from each shaft, a separate pile being made of each grade of ore.

Generally the floor of the stock-pile is five to seven feet above the railway level, and the side next the railway is supported either by a rough stone-wall, as at the Cleveland, or by crib-work, as at the Champion. In some cases, as at the New York, piles are made on the general level.

In the shipping months, June, July, August, September, the stock-pile is gradually removed. At first the ore is shovelled directly into the cars, then temporary narrow-guage tracks are laid, and ore carried in small tram-cars to the railroad, enough tracks being laid to have one for each railroad car being loaded.

RAILROAD TRANSPORTATION AND TRANS-SHIPMENT OF ORE.

The ore from the different mines is carried by the Marquette, Houghton, and Ontonagon Railroad to L'Anse or to Marquette on Lake Superior, or by the Chicago and Northwestern Railroad to Escanaba on Lake Michigan. At these points the ore is loaded on sailing-vessels and steamers, and carried through the lakes to Cleveland and other eastern ports.

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At each of the shipping-points are ore-docks for facilitating the trans-shipment of the ore. These docks are built on trestle-work, high above the water, and are provided with a series of ore-pockets into which the ore is dumped from the cars, railroad tracks being laid over pockets the whole length of the dock. Long, hinged chutes lined with iron serve to conduct the ore from the pockets to the deck or hold of the vessel.

The ore-cars, having a capacity of about eight and one-half tons, are provided with movable bottom doors by which they are discharged. The pockets have a capacity of from 50 to 100 tons each. The dock at Marquette has a total length of 1,222½ feet, a working length of 720 feet, is 53 feet in width, with four tracks, and rises 38 feet above the water.

The foundation is of piles cut off three feet above the water and capped; above the caps are placed the sills, and upon these rest the main *bents* constituting the dock-support. There are 136 pockets on both sides of the dock, of which 120 (vessel-pockets) have a capacity of 55 tons each, and 16 (steamboat-pockets) of 100 tons each.

The bottoms of the majority of the pockets have a slope of 39°, and are lined with ½-inch iron plates 12 inches square. The mouth of the lowest pocket is 17 feet above the water, that of the highest, 22 feet. The ore is delivered on board the vessels by means of spouts hinged at the mouths of the pockets and raised or lowered by a crab-wrench placed above. The spouts are lined with ¼-inch iron plate.

The dock is of sufficient capacity to load eight vessels at the same time. The vessels average about 650 tons, and with that tonnage may be loaded in one hour and thirty minutes.

The estimated capacity of the dock is 500,000 tons of ore for the entire season of navigation.

The ore-docks at Escanaba and L'Anse are similar to those at Marquette. The dock at Escanaba is upward of 1,300 feet in length, 32 feet in height, 37 feet in width, and has 352 pockets, having a total capacity of 24,000 tons.

The L'Anse dock has a length of 546 feet, is 36 feet wide, and 38 feet high. There are in all 84 pockets with an average capacity of 75 tons each. The arrangements for loading and discharging are in every respect similar to those described, the spout when lowered forming a continuation of the pocket-bottom and delivering the ore directly into the hold of the vessel.

(To be continued.)

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MINES OF THE APPALACHIAN RANGE.[1]

By GEORGE B. HANNA, E.M. ('68).

NORTH CAROLINA.

ON the central belt of North Carolina, comprising eleven counties in part, and covering a territory one hundred miles long, and, on an average, fifteen miles wide, vein-mining only is undertaken. The placers have long since been exhausted, with the exception of very small and unfavorably lying patches, which admit of unimportant operations only.

In Guilford County the mines at work are the Fentress and the Gardner Hill, but the operations are on a small scale. The Conrad Hill is in process of reopening.

The Allen and the Eureka, near Thomasville, Davidson County, have also been worked to a small extent, and at the Silver Valley Mine, near the same place, the operations are confined to developments, and there seems at present little disposition to mill the ore, although a 10-stamp battery is at hand ready for work.

The Silver Hill Mine, near Lexington, has been one of the most extensively worked mines in the State, and, with one exception, is the deepest. The ore is a refractory galena and blende, with a little silver and a trace of gold.

About Salisbury, in Rowan County, are numerous mines, which would doubtless be worked if the conditions of the metallurgical industry of this section afforded a reliable market for ores.

At the Dunn Mountain Mine, three miles east of Salisbury, the company have been content to develop before erecting mills. This ore has considerable iron pyrites and some copper pyrites, but is, for the most part, brown oxide of iron with quartz and slate.

The Gold Hill Mine, fifteen miles southeast of Salisbury, although not on the central belt, is so intimately connected with it as to claim attention here. This has been the most ex-

[1] Extracts from a paper by Mr. Hanna, in the Report of the Director of the Mint, Washington, 1881, pp. 168 et seq.

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tensively worked and most noted mine in the South, having reached a depth of seven hundred and thirty-five feet. It is not worked now, except in a petty way, and chiefly by a retreatment of the old tailings.

The operations of this section cannot be dismissed without an allusion to the Davis and Tyson metallurgical works, situated two miles south of Salisbury. A large plant, covering at least one-fourth of an acre, has been erected, and supposed to be capable of treating ten tons per day. The method is essentially one of chlorination.

Chlorination works have recently been erected at Mount Pleasant, Cabarrus County, and the superintendent states that they have been successful.

With a very few exceptions, the best known mines are situated in Mecklenburg County, but three mines only are in operation.

The Rudeshil, one mile south of Charlotte, is credited with the production of $1,000,000. At present the depth is little more than two hundred feet, and the lowest level one hundred and ninety feet. This mine is capable of turning out large bodies of ore suitable for milling, and a fine grade of sulphurets adapted to a smelting treatment, of which latter class a considerable quantity has been shipped to Northern smelting-works.

The New York and North Carolina Reduction Works, located at Charlotte, commenced operations about six months ago, but the amount of work done to this time has not been large, as the supply of ore has not come in rapidly.

On the western belt of North Carolina the King's Mountain Mine, in Gaston County, is the only mine now worked to any important extent. This mine has been very productive. It is worked at a depth of three hundred and twenty feet. The ore is impure limestone, with a small percentage of galena, blende, iron and copper pyrites, and other rarer minerals[1] in very small proportions. There are two veins, or ore-bodies, the front one being worked at present. This ranges from eleven to fifteen feet in thickness. The whole material is low grade, but as it is cheaply mined and easily milled, and has but a trifling proportion of refractory sulphurets to lessen the yield of gold, it can be worked at a profit at figures which

[1] Native gold, altaite, nagyagite, etc.

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with most kinds of ore would barely suffice for mining and hoisting. It is provided with four double batteries and other machinery capable of treating sixty to eighty tons per day.

The "gravel" mines of this western section have attained a great celebrity. The most noted area is where Burke, McDowell, and Rutherford Counties come together. An exact survey would probably show 250 square miles of these deposits. As the processes are made more economical and efficient, the paying areas will be more numerous. The gold hitherto extracted has been from the gravel exclusively. Its ultimate source has never been satisfactorily ascertained, but as rich gold-bearing quartz has often been found with the gravel, it apparently came from the innumerable seams of quartz which everywhere traverse this section. This locality, in common with the whole of the Carolinas and Georgia, has undergone extensive alterations and decomposition, and subsequent wearing down, by which the gold has been left in the surface-soil, and more largely in the bottoms of streams both ancient and modern. The greater part of these accessible places was long ago exhausted, and the work of the future will be on the deep-lying gravels, which require expensive digging to remove the surface-soil, or else a heavy and powerful stream of water to wash it away. The product of this area is probably not less than $30,000 per year, and, in seasons when water is abundant, will exceed this.

In the eastern belt of North Carolina the formation is slate, sometimes argillaceous, sometimes talcose and chloritic, sometimes micaceous, and occasionally hornblendic. At present but few veins are worked; the chief localities operated are in "gravel."

The formation at the Jones and Keystone Mine, fifteen miles nearly east of Thomasville, is most curious. The tracts owned or controlled by the company comprise 293½ acres. The ordinary talcose and chloritic schist of the section is here highly decomposed to a depth of 20 to 40 feet, and everywhere stained with peroxide of iron from decomposed sulphurets. From many assays made from samples taken by myself from many and widely separated points, I think most of this material contains gold to the extent of $2.50 per ton, with chimneys carrying ore of much higher grade. The material is soft, and easily crushed by the hands, and mining is

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easy and cheap; and as the conformation of the surface allows the use of gravity tramroads, the ore can be put in the mill-house at an expense of a few cents a ton. The supply of water is very scanty, and this lack has seriously hindered work, otherwise the property would be very valuable. At present work is suspended to allow the erection of a plant to treat the ore by a recently invented French process.

The Beaver Dam Mine, in Montgomery County, is thirty miles east of Salisbury. The property comprises 800 acres, of which nearly one-half is claimed to be underlaid by gravel. It is worked exclusively as a hydraulic mine, and on a system of hydraulic mining peculiar to the South, the water being thrown directly on the gravel by a steam pump, the supply of water being obtained from the creek.

The Sam Christian Mine, in the same county, has had a famous history, and the present operations show that its resources are far from exhaustion. The place is worked only as a hydraulic mine. At present two old streams are in process of uncovering. A curious fact about this mine is the absence of gold in small grains, and its occurrence in nuggets.

Any effort to estimate the present production of North Carolina is difficult, as several producers of bullion have shown a reticence in answering questions. The records of the assay office at Charlotte show deposits amounting to $57,092.14 in 1880, but the entire product of the State did not pass through that office. As regards the production in 1881 another source will be added, viz., metallurgical works; but I do not anticipate any large addition from this species of work, as the establishments are new, and the methods as yet to be fully applied and perfected.

SOUTH CAROLINA.

Most of the general conditions mentioned under mining in North Carolina, apply, with equal force, to South Carolina, for most of the formations are the same, and the same industrial conditions exist.

Vein-mining is conducted at only two places, most of the work being of a hydraulic character. The work is not so extended as in North Carolina.

The Brewer Mine is located in Chesterfield County, twenty-nine miles nearly south from Munroe. In this mine there is no vein in the usual sense of the word; at present only

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"gravel" is worked, and geological changes have distributed this more or less through the surface-soil, and on the bed-rock of old streams. The recent work has yielded $1 per square yard of bed-rock uncovered. There is another curious body of auriferous material on this property which has proved a puzzle to geologists. I refer to an enormous mass of fine white or slightly stained sand, so loosely aggregated that much of it can be easily pulverized by the fingers. It is traversed irregularly by masses of horn-stone or agatized quartz, bearing fine iron pyrites, etc. This material is everywhere auriferous. It has been worked in an irregular way for thirty-eight years, and is pitted with old shafts. No determination of the limits of this body has as yet been made, but from the outcrops found at widely distant points, there must be several hundred acres. The larger part of this body is admirably fitted for hydraulic mining. The supply of water is the most important consideration here. The whole mine being so high above the surrounding country, the sole means of supplying it is by pumping apparatus. A powerful establishment is now in operation, capable of supplying one thousand gallons per minute, and thought to be capable of uncovering one hundred square yards of bed-rock per day. A return of $18,000 to $20,000 may reasonably be expected during the present year.

The Haile Mine is situated in Lancaster County, ten miles west of the Brewer. The property is one vast bed of talco-chloritic and micaceous schist, with alternations of siliceous schist. The ore-masses are simply the schists of the country more highly charged with mineral matter; they are lenticular in shape, with a width varying from six to sixty feet, and alternating with heavy bodies of nearly pure iron pyrites, sometimes eight feet thick. The average value of the ore is thought by the assayer of the company to be $7 per ton for the schists, and not less than $15 per ton for the heavy and pure sulphurets.

At the Dorn Mine in Abbeville County nothing of importance is now done. This property was once worked with a large return, and it is claimed that there are still large bodies of ore in the mines.

The production of South Carolina for 1880 has been very small; the total amount deposited at the Charlotte assay office was $8,082.58. The total production may have been $12,000. The yield of 1881 will be much larger.

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GEORGIA.

The auriferous area of Georgia covers something more than one-third of the northern portion of the State, and stretches from South Carolina on the east to Alabama on the west; it is geologically the extension of the area of North and South Carolina.

The occurrence of the gold is not very dissimilar to that of Western North Carolina. Schists everywhere make up the country—chlorotic, micaceous, talcose, and sometimes, though rarely, hornblendic. They are usually decomposed to a great depth and easily broken up, and when not decomposed they are soft enough to yield to the pick.

The gold appears to occur in seams of quartz, which traverse the strata in great numbers, usually parallel to them, and varying in thickness from one-fourth of an inch to six inches, rarely more. There are zones of strata in which these seams are particularly abundant. The mining is principally carried on in these zones, as outside of them the seams are so few as to make the ore of too low grade to be of economic importance. The ore is of very low grade. I could not learn that any effort has been made to ascertain exactly the average value of the ores, but, from a comparison of all the information attainable, I am inclined to put it at not more than $1.25 to $1.50 per ton.

The general method of exploiting is as follows: The mine being selected, the first care is to secure water from some ditch company, and draw it into their own reservoir at such an elevation as will allow it to command every part of the ore-body. Now and then this is found impracticable, and the water is pumped from the highest attainable point of flowage, up to the desired point. At the same time the strata which overlie the body of ore are removed or opened so as to expose the ore, and to allow of its ready removal through the cut to the mill, which is established in some ravine near at hand.

At the outset the treatment is purely hydraulic. The water is thrown against the ore with the tremendous force due to its fall of sixty or one hundred and fifty feet, or more, and rapidly eats it out of its lead, and washes it into sluices in the bottom of the cut, and down into the mill-house. Sometimes a few blows from a sledge-hammer are needed to break up the harder material, but, generally, by the time the material has

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reached the mill, which may be from one to twelve hundred yards away, the decomposed and soft slate is well pulverized, and much of it floats away, leaving the gold in the sluices, while the harder slate and quartz is carried into the mill, pulverized and amalgamated.

The mills are of the usual Californian pattern, modified to suit the ore. They are generally run in double batteries of ten stamps, of three hundred and fifty to four hundred pounds, making sixty-eight-inch drops per minute, and discharging through screens with meshes of $\frac{8}{32}$ to $\frac{1}{4}$ of an inch opening. The softness of the material, and the coarseness of the screen allow the treatment of about three tons per day per stamp, which corresponds with five to ten tons of material as it stood in the mass. Almost no hauling is given this material, so that twenty-five men, as in the Pigeon Roost Mine, will supply the ore, and do all the work for a forty-stamp mill. The cost of water is the most serious item, the charge at present being twelve cents per day per inch, of which rarely less than twenty-five inches are used.

In Lumpkin County the two most prominent mines are the Pigeon Roost and the Findley, and at both the production has been large. The former, in particular, is especially well opened. It is operated by a turbine, so that the cost of motive power is extremely small. The yield in 1879 was $35,000. The Findley Mine is also well opened, and provided with ample mining ground to supply its large mills, of which it has two, provided with fifty stamps in all.

In Dawson County a few mines are now opening, but the work of production at present is confined to individual miners. A similar statement may be made regarding Hall and White Counties.

The production of Georgia for 1880 is estimated at $121,880.

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PROFESSIONAL NOTES.

Mining Engineering.

Experiments and Improvements in the Mines of Prussia during 1880 *(Continued).—Automatic Safety-Gates.*—Numerous devices have been tried at the Saarbrucken Coal Mines for the automatic raising and lowering of the safety-gates at the mouths of the shafts, with a view to obtain a form of gate of simple construction, which will work with certainty, with little noise, and without hurtful shocks to the cage or hoisting ropes. At the Altenwald Shaft, No. 1, the gate is raised by means of a long lever, pivoted at the middle, one end of which is struck by the ascending cage, while a rope attached to the other end, and passing over pulleys, raises the gate. When the gate has thus been raised to the required height, it is held in position by a latch, which hooks into an eccentric cam on the axis of the lifting lever. On lowering the cage, this latch is released by means of a second lever below, which is struck by the cage in descending, thus permitting the safety-gates to fall by their own weight and close the mouth of the shaft. The fall of the gate is checked and controlled by a piston attached to the lifting lever, which works in an air-cylinder. This device has proved successful and quite satisfactory in all respects.

A simpler form of safety-gate is that which is raised and lowered directly by the cage, without the intervention of levers or ropes. The chief objection to this form is the noise and shocks attending the opening and closing of the gates, which are struck violently by the ascending cage and dropped with some force when the cage descends. This difficulty has been overcome by the use of a light lattice gate of steel, weighing less than fifty pounds, running in guides and with projecting arms provided with rubber buffers which are struck by the ascending cage. When the cage is lowered, the gate falls on spiral steel springs. With these light gates not only is the noise much lessened, but the hurtful shocks and consequent strain on the hoisting ropes and connections are much lessened.

Safety-gates at the bottom of the shafts or elevators are sometimes desirable. At the Gerhard-Prinz-Wilhelm Mine, a gate at the bottom of a surface-hoist is raised by means of two long levers pivoted at the middle, one on each side of the shaft. The outer ends of these levers are attached to the gate, while the inner ends are arranged to be struck and depressed by the descending cage, which thus raises the gate on reaching the bottom of the hoist.

Attachment of Wire-Rope to Cage.—At one of the Saarbrucken Coal Mines the ropes pass through or between the top bar or bars of the cage and through a rubber buffer, and is secured below by means of four small clamps attached to the rope.

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Tail-Rope under Hoisting Cages.—At the Cons. Myslowitz Mine, in Upper Silesia, the hoisting drum is replaced by a sheave at the top of the shaft-house, with an oak-lined groove. Brake-blocks, three in number, grip in the same groove. Two cages are used, attached to the ends of a wire-rope passing over the sheave. The rope is of such a length that when one cage is at the top, the other shall be at the bottom of the shaft. To balance the weight of the hoisting-rope, which is now all on one side and again all on the other, a loop of old rope, of equal length, is attached to the bottom of the two cages.

Hoisting with Endless Rope (Köpe System).—A further improvement on the tail-rope system is used in a winze in the *Maria* Coal Mine in the Wurm mining district. Two cages are used, each arranged for two cars, side by side, each car containing 10 cwt. of coal. The endless rope passes through wrought-iron tubes in the middle of each of the cages, and is secured below the cage by wrought-iron clamps. By shifting the clamps, the distance or length of rope between the cages can be altered at will, thus making it possible to hoist, at different times, from different levels, without foregoing the advantage and economy of balanced cages. The endless rope rests in a deep groove of the hoisting-sheave, which is driven by a two-cylinder compressed-air engine working expansively, with Farcot's cut-off. Cylinders 13½ inches diameter, with 31½ inches stroke. Maximum air-pressure, five atmospheres (now working at three atmospheres). Hoists 200 tons in 12 hours to height of 886 feet.

The advantages claimed for this system are: 1st, machinery requires but little room and less expensive foundations than if a drum were used; 2d, the height of the sheave above the landing stage need not be as great as in the ordinary plan; 3d, the wear of the rope is less, as it is subjected to but one bending; and 4th, the shifting of the cages is easy and simple, and permits the wear of the rope to be equalized.

The main disadvantage lies in the fact that any breakage results in the loss of both cages.

Telephones are in use at the shafts of the Cologne Mining Company at Altenessen. Siemens's telephone and call, working without a battery, is used, and underground stations are proposed. An underground line is in successful operation at the Königen-Louise Mine in Upper Silesia.

Surface Transportation—Narrow-Gauge Railroads.—Two narrow-gauge (30-inch) roads have been constructed in the Mansfield copper district for the transport of the copper schists, respectively 0.6 mile and 3.1 miles in length.

Wire Ropeways.—Bleichert's elevated ropeway is coming into more general use for transport of minerals between the mines and the railway stations, etc. In this system of rope-transportation, the buckets are attached to hangers, with grooved wheels, which travel on two parallel lines of fixed wire-rope, or, more recently, of round iron rods as tracks, the buckets being attached at intervals to an endless driving-rope. This endless rope passes around horizontal sheaves, at one end

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of the line provided with driving gear or brakes, as the case may require; and at the other end with some arrangement for keeping the driving-rope taut.

At the Mont Cenis Mine, near Dortmund, a self-acting wire ropeway, 290 feet long and 45 feet higher at one end, is used to convey tailings and waste rock from one of the mines to a dumping-place.

A ropeway has been constructed to connect the Briloner-Eisenberg Iron Mine with a railroad station at Olsberg, 2.2 miles away. The cost of the road, including compensation to land-owners, was about $20,000. The capacity of the road is about 90 tons per day, and the cost of transportation is 7.2 cents per ton, or about one-fifth what it formerly cost.

A ropeway on the same system is in use at the Beilstein Iron Mine in the Dillenberg mining district, carrying ore from the mine to the railroad, and bringing back coal for the boilers.

At the Mathias Zinc Mine, in Upper Silesia, a ropeway two-thirds of a mile in length has been constructed, which runs in a straight line to the Upper Silesia narrow-gauge railroad. The cost of transportation, at the rate of 50 tons per day, is 9.7 cents per ton.

A ropeway, nearly 3 miles in length, in two sections, has been built between the coal mines of the Cons. Johann Baptista Company, in Lower Silesia, and the Dittersbach-Glatz Railroad. This ropeway not only crosses a deep valley, but makes at the junction of the two sections an angle (qy. vertical angle?) of 150°. At this point a 10 H. P. driving engine supplies the motive power. The round iron rods forming the track are here, for a distance of about 55 yards, respectively $1\frac{1}{8}$ and 1 inch in diameter for the full and empty lines, and will sustain loads of 5 and $3\frac{1}{2}$ tons. The lengths of rod are welded together, and, at distances of 110 yards, are joined by screw muff-couplings of steel. Where the line crosses the valley a wire-rope is substituted for the iron rods. The driving-rope is of steel, three-fifths of an inch in diameter. The buckets have a capacity of about 10 cubic feet (or $8\frac{1}{2}$ bushels of coal), or nearly 700 pounds, and are attached to the driving-rope at intervals of 55 yards. The capacity of the line is 240 tons per ten hours. The cost was about $27,500, and, including all expenses and an allowance of 10 per cent. for a sinking fund, the cost of transportation is about 15 cents per ton.

A Rack-Road on the Riggenbach System has been constructed at the Friedrichssegen Mine. The grade of the road follows closely the natural surface of the ground, thus lessening the cost of construction, and is built in part as an ordinary adhesion road, the rack being used only on the steepest grades. The total grade in a distance about one and one-half miles is 384 feet, giving an average grade of 1 in 21.4. The ruling grade on the adhesion road is 1 in 20.5, and on the rack-road uniformly 1 in 10. There is about three-eighths of a mile of rack-road in three sections, respectively 88 yards, 390 yards, and 182 yards in length. The rack is made of two angle-bars, with cross-pieces of trapezoidal section forming the teeth, and weighs about 110 pounds per yard; teeth, 4 inches apart from centre to centre; diameter of spur-wheel, 1 metre; curves in rack-roads, 160 yards radius. To insure the exact interlocking of the spur-wheel in the rack when the locomotive passes from the ordinary road to the rack-road, there is placed at

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the beginning of each section a length of rack supported on heavy spiral springs, with teeth slightly nearer together than the regular distance.

The locomotive differs from the ordinary type in that the power is transmitted to the spur-wheel axis by gearing, from which axis motion is transmitted to the drivers by connecting rods. The spur-wheel is placed high enough not to strike the rails between the tracks on switches and crossings, and consequently the rack must also be placed at a corresponding height above the level of the rails. The engine can haul 10 tons up the grade of 1 to 20 of the ordinary track, or the same load on the 1 to 10 grade of the rack-road.

The cars are side-dumping, built of iron, holding one cubic metre, and weighing one ton empty. In some respects their construction is unique, the chief peculiarity being in the shape of the body, which has a curved bottom and flaring sides, so shaped as to facilitate the dumping of the load. In dumping, the car-body rolls over sideways, the curve of the bottom being of such form that the position of the centre of gravity remains at about the same level.

The highest speed on the ordinary track is six and three-fourths miles per hour, and on the rack-road one-half this. Cost of the road, less than $20,000 per mile.

The Electric Light is now used to light pit-banks, screens, coal-docks, and the surface workings.

An Illuminating Stove is also employed for the same purpose, consisting of a portable stove with a 16-inch step grate, surmounted by a mammoth argand burner, of simple construction. With about 160 pounds of coal (bituminous) it will give a bright flame, 7 inches in diameter and about a yard in height, lasting for two hours.

Ventilating-Machines.—Recent tests of the Guibal fan give results which are not altogether satisfactory, as showing a low degree of efficiency. The Pelzer ventilator, an exhaust fan with screw-formed blades on a conical base, shows a high degree of efficiency, and has otherwise proved satisfactory. Detailed drawings of this fan are given in the paper. Hand ventilators on this plan of construction have proved very successful. A fan for ventilating headings and places where a temporary or special air-current is needed is also described and figured. The chief peculiarity is in the motor, which consists of a small reaction wheel, driven by water-power, and fixed on the axis of the fan itself.

ORE-DRESSING.

Mechanical Assays for the Control of Coal-Washing.—Heretofore there have been but two methods of controlling the washing of coal, one by the inspection of the coal and waste, and the estimation of the amount of slate left in the washed coal and the amount of good coal lost in the waste, and the other by the making of frequent ash-determinations in samples of coal and waste. On account of the presence of bituminous shale, and of so-called slate, or slate with varying percentages of carbon, both methods fail to give satisfactory results. By

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inspection, it is often difficult to distinguish such bituminous shales from pure coal, even when they contain a large percentage of ash. The difference in specific gravity is not enough to permit the separation of these bituminous shales by vanning in a spoon, pan, or shovel, as in the tests made for the control of metal-dressing. Finally, while the determination of ash in the washed coal serves to indicate its purity, a similar determination, by ignition, of the amount of carbon in the waste by no means serves to indicate the amount of good coal that has been lost. For example, two samples of waste from the washing of a coal showed 48 and 45 per cent. of ash respectively. A mechanical assay, by the method about to be described, showed in the first 25 per cent. of good coal, and in the other 12 per cent. only.

Mr. Aug. Erich, a Bohemian mining engineer, has recently published [1] a method for the assay of the products of coal-washing and for the testing of coals, to determine whether they can be washed with profit. He uses a series of solutions of zinc chloride of different specific gravities: 1.8, 1.7, 1.6, 1.5, 1.4, 1.35, 1.3, and 1.27 respectively. By means of these solutions, the coal is divided into nine portions of different specific gravity. A determination of the ash in each of these at once shows where the line can most profitably be drawn between coal and waste, how far it will be possible to purify the coal by washing, and the approximate loss which will result from washing to any given degree of purity.

The results of a large number of assays are given in Mr. Erich's paper, from which we select the following tests of different samples of the same coal as types:

Specific Gravity.	Before Washing.			Badly Washed.			Washed Coal.		
	a.	*b.*	*c.*	*a.*	*b.*	*c.*	*a.*	*b.*	*c.*
Above 1.80	10.63	80.5	8.55	4.59	69.5	3.19	..	..	..
1.70 to 1.80	3.78	43.8	1.65	0.72	47.3	0.34	..	..	..
1.60 to 1.70	0.40	35 2	0.14	0.93	34·5	0.32	..	..	..
1.50 to 1.60	1.33	22.0	0.29	1.13	28.4	0.32	1.20	20.0	0 24
1.40 to 1.50	0.96	16.1	0.15	1.44	13.1	0.19	2.52	13.5	0.34
1.35 to 1.40	5.12	9.4	0.47	5.77	11.7	0.67	4 61	11.2	0.51
1.30 to 1.35	6.83	4·5	0.30	10.94	4 6	0 50	27.58	4·7	1.29
1.27 to 1.30	61.05	1.6	0.98	66.29	1.0	0.66	60.98	2.1	1.28
Below 1.27	6.68	0.5	0.03	7.01	0.5	0.03	3.06	0.6	0.02
Slime	3.22	7.3	0.23	1.18	10.3	0.12	0.05	7.5	0.00
Total	100.00	..	12.79	100 00	..	6.34	100.00	..	3.68

Columns marked *a* in the above table show the percentage of material, coal or shale, corresponding to the different specific gravities, found in the sample tested. In columns *b* are the results of the ash-determinations in each portion so separated. Finally, the figures in columns *c*, obtained by multiplying *a* by *b*, show the amount of the total ash chargeable to each constituent part of the coal, the total, in columns *c*, being the total percentage of ash in the sample.

For controlling the operations of the wash-house, Mr. Erich uses a single solution of such specific gravity that the pure coal will float and

[1] Berg und Huttenmannischer Zeitung, vol. xl., pp. 473 and 487, December 16 and 23, 1881.

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the bituminous shale and slate will sink. The specific gravity of this solution will, of course, vary with the character of the coal, and will be determined by assay, but for general purposes a specific gravity of 1.5 has been found most applicable. With such a solution the foreman can test the coal and waste from the different jigs, and estimate closely the amount of coal which is contained in the waste, or the slate in the washed coal. Such control is especially valuable when the amount of ash allowed is fixed by contract.

The method was suggested by a paper by V. Goldschmidt[1]: "On the Separation of the Constituent Portions of a Mineral according to their Specific Gravities by Solutions of High Specific Gravity."

Having experimented with various fluids, Erich concludes that zinc chloride is the most appropriate, as it may be evaporated to 1.9 specific gravity without becoming too thick. Being somewhat expensive, it is used for work in the laboratory only, and is mixed with calcium chloride solution for work on a larger scale, as in a washer. All other salts are either too expensive, as bromides, iodides, etc., or of too low a specific gravity, as magnesium chloride; while acids and other fuming or chemically decomposing agents are naturally inapplicable.

The chloride is prepared by dissolving zinc scraps, in excess, in crude hydrochloric acid, filtering the solution and evaporating to the required specific gravity.

Usually it contains a certain amount of iron, but this does not affect the ultimate result to any very appreciable extent, and iron is precipitated as oxide after the solution has been used several times. It may be precipitated out at once, however, by passing chlorine and then adding zinc oxide, and filtering through asbestos or sand. Prepared in this manner, a litre costs about 0.20 mark, or about 5 cents.

The specific gravity of the liquid can be determined with sufficient accuracy with an aërometer, and the different solutions necessary for the laboratory tests brought to the proper specific gravity by the addition of water. In diluting, care should be taken to stir the solution well, to avoid the formation of basic salts.

The operations are performed in beakers for fine, and in tubs for very coarse grains of coal. The beakers used for accurate work contain from 500 to 1,000 c.c. The author prefers glass cylinders, 20 ctm. high and 8 ctm. in diameter, as being less fragile than beakers.

The coal is first washed to remove the slime, which may be dried and the ash determined, and then dropped into the heaviest solution. That portion of the coal which is heavier than the solution at once sinks to the bottom. The portion which remains floating on the liquid, is then removed by a spoon of 100 mesh wire gauze, drained thoroughly, and placed in the next heaviest solution, and the same operation being performed for the eight beakers or cylinders, one more product is produced than the number of solutions used. The sediments are caught on a skimmer, washed by decantation, dried, and weighed, and finally ignited and the ash determined.

The weight of coal used, is from 500 to 1000 grammes. For fine grains and slimes, much less. The time employed for coarser varieties (500 grammes, in four beakers), is from ten to twelve minutes; for

[1] Mr. Goldschmidt used a series of solutions of 1 part KI to 1.24 Hg I_2 in water, with specific gravities of 3.196 to 1.0.—Neues Jahrb. f. Min. Beil., B. 1, p. 183.

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fine grains from a quarter to half an hour. For one solution, two to three minutes, excluding weighing and drying.

The assay of slimes is most conveniently made in a burette, a single solution of mean density being used, and the sediment being drawn off at the bottom by means of a stopcock or pinchcock with an opening of sufficient size.

In making tests of tailings from jigs, or of the washed coal, the size of the grains of coal, and the proportion of undersized material in the waste, or of slate in the coal should be noted, as this will indicate whether the preliminary sizing of the coal has been properly effected.

While it is found that, as a general rule, the amount of ash increases in direct proportion to the density, still this is not always the case, owing to the presence of included gases on the one hand, or heavy minerals on the other, such as blende, galena, pyrites, etc., which affect the specific gravity of the coal.

This method of assay is not applicable to ores, as solutions of the required specific gravity (2.0 to 5.0) cannot be obtained. A melted mixture of platinum chloride and zinc chloride, proposed by Bréon (*Comptes Rendus*, xc., p. 626), though perhaps useful for scientific experiments would be inconvenient for trade work. K. N.

Civil Engineering.—Progress Abroad during 1881.

The Bavarian Government has been conducting a series of interesting experiments on the resistance of curves to motion on railways. The principal conclusions so far are as follows: Rolling-stock of the same construction and weight afford resistances varying to a very considerable extent, due to irregularities and the amount of wear. The coefficient of resistance diminishes with increasing load. Locomotives have a higher coefficient of resistance. The resistance of all rolling-stock increases with the speed, this increase being greater than in direct proportion. For a straight track the following empirical formulas were determined: for cars, $\rho = 0.0025 + 0.00000021V^2$; for locomotives, $\rho = 0.0050 + 0.00000021V^2$, ρ being the coefficient of resistance, and V the velocity of motion in kilometres per hour. If the increase of resistance, due to a curve, is considered as an additional resistance over and above the constant resistance of a straight track, as determined by the above formulas, this additional resistance is independent of the velocity of motion. For the coefficient of the average additional resistance due to a curve: $w = \frac{0.6504}{R - 55}$, R being the radius of the curve in metres. A narrowing of the gauge by half or more unexpectedly increased the resistance. The additional resistance due to a curve of 150 metres radius was diminished 43 per cent. by wetting the rails with water.

The German Government possessed, January 1, 1880, about 188,480 miles of main-line railroad, of which 6,059 miles were double track; of branch roads 1,590 miles were in operation, of which 80 miles were narrow-gauge.

The Railroad, or Inclined Plane, on the slope of Mount Vesuvius

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was opened for traffic, June, 1880. The chief difficulty in construction was the unstable nature of the slope, being chiefly loose sand, pebbles, scoriæ, etc. The track is laid on longitudinal timbers, supported by ties and diagonal braces. Sliding is prevented by walls built some distance apart. There are two cars, holding twelve persons each, and built in steps, one descending while the other ascends.

It is proposed to dig a canal from Paris to the Departments du Nord and Pas-de-Calais to develop the coal-deposits of those counties, their production for 1878 having been seven million tons, or two-fifths of the entire production of France. Its construction will involve a tunnel 4,544 metres long; its total length being about 116 miles, of which 30 miles are taken from other canals. The canal is to be 2 metres deep and can be deepened to 2.5 metres. The approximate cost is estimated at $21,000,000. A tonnage of four millions is expected, which will place the freight at from 37 cents to 50 cents.

It is also proposed to build a canal through Southern France to connect the Atlantic Ocean with the Mediterranean. The depth is to be 8.5 metres. The locks double, one of 11 metres and the other of 25 metres breadth and 150 metres long.

The annual receipts of the Suez Canal are about $3,400,000, of which one million is spent on repairs. The original outlay was $94,500,000. K. N.

Tripolith, a substance sold in Germany as a cement, has been found, by Petersen, to consist of an impure sulphate of lime mixed with carbon, about 75 per cent. of the former to 11.5 per cent. of the latter, the remainder being carbonates of lime and magnesia—about 4 per cent. of each, together with small amounts of sand, oxide of iron, water, etc. Tests made with it showed it to afford a cement having but about one-third the strength of Portland cement, and, moreover, in the course of a few months it lost strength.—*Ber. Deuts. Chem. Gesell.*, 1881, p. 2361. E. W.

Analytical and Applied Chemistry.

On Filtration with Bunsen Pump.—An excellent substitute for the expensive platinum cone generally used for rapid filtering at the Bunsen pump, is a piece of strong linen cut in the shape of an ordinary filter-paper. It has the advantage of fitting snugly into the funnel, and of filtering much more rapidly. It is in universal use in Prof. Hofmann's laboratory. P. N.

Reduction of Copper-Foil for Nitrogen Determinations.—The tedious and expensive operation of reducing copper-foil in a current of hydrogen, is avoided by heating the foil in a long flame, and, while red hot, dropping it into a test-tube containing two or three drops of methylic alcohol. The foil is then dried at 100° in the test-tube.

Percy Neymann, Ph.B.

Berlin, January 6, 1882.

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For Volumetric Determination of Chromium and Manganese in ores, E. Donath uses a strongly alkaline solution of permanganate. The reaction on a hot solution of Cr_2O_3 and MnO compounds in the case of chromium is then:

$$K_2Mn_2O_8+Cr_2(HO)_6+2\ KHO=2\ K_2CrO_4+2\ MnO_2+4\ H_2O.$$

For manganese it is:

$$3\ MnSO_4+K_2Mn_2O_8+4\ KHO=5\ MnO_2+3\ K_2SO_4+2\ H_2O.$$

The presence of ferric iron or of alumina does not interfere. The end reaction is of course the same as in the titration of iron by Marguerite's process.—*Ber. Deuts. Chem. Gesell.*, 1881, p. 982.

Assay of Cupriferous Blende.—The *Chemical News* gives the following method for assaying cupriferous blende by R. Monger, of Swansea; ½ grm. of the blende is taken and treated with aqua regia and evaporated; it is remoistened with hydrochloric acid, 10 to 20 c.c. of ammonia added, and the iron filtered off and washed. The filtrate, which is now about 200 to 250 c.c. in volume, is acidified with hydrochloric acid, heated, and placed in a porcelain dish ready for buretting with ferrocyanide of 0.01 strength. To the solution of zinc and copper, enough sodic sulphide solution is added to precipitate the whole of the copper and leaving a little excess; then the buretting with ferrocyanide, and uranium acetate as indicator, is proceeded with. In some experiments, he took a non-cupriferous blende and weighed out six samples into three, to which some sulphate of copper solution equal to two and a half per cent. copper in the blende was put. Proceeding with them as above explained, perfectly concordant results were obtained.—*Engineering and Mining Journal.*

Separation of Lead Sulphate and Silver Salts from Silicic Acid.—Very frequently in the analyses of silver and lead ores, the chemist will have cases presented where the percentage of cerargyrite and anglesite is quite large. This is especially true in connection with lead-smelting works, now quite numerous in Colorado, Utah, and Nevada. In most cases, for the control of the smelting operations a determination of the percentage of iron and silicic acid will be found sufficient. In large works the chemist may be called upon to make from six to twelve silica determinations per day, owing to the many kinds and the amount of ore to be treated; which amount will vary from seventy-five to two hundred tons in twenty-four hours. It will therefore be seen that rapidity, combined with accuracy, is the great desideratum.

There has been published, however, no good and practical method for the elimination of large quantities of the sulphates of lead and of silver salts from silicic acid. We are therefore led to detail the mode of procedure we have used for a long time, and with most excellent results.

Take one-half to one gramme of ore from the "Assay Sample" and weigh out first, roughly, the requisite amount. Inasmuch as the assay sample (representing an average of from twenty-five to two hundred tons ore) has only a sufficient fineness to pass through an eighty-mesh sieve, we transfer the amount roughly weighed out, directly to an agate mortar, and finely powder. Next weigh accurately the amount to be used, and transfer to a glass-covered casserole. Pour upon the ore 15 to 25 c.c. concentrated hydrochloric acid; place

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upon a water-bath, and, at the expiration of from three to five minutes, the carbon dioxide and chlorine (in cases of manganese ores) will be expelled, and no violent action ensues—the portions adhering to the glass cover are washed into the casserole, and the cover removed. Evaporate to dryness. Generally the first evaporation will be found sufficient, and sometimes it is not necessary to evaporate at all, when no soluble silica occurs. After evaporating, add about 10 c.c. concentrated hydrochloric acid, warm, and then add hot water until three-fourths of the casserole is filled, cover, and add directly one or two pieces of zinc. We have now a number of chemical reactions ensuing, which number, of course, varies according to the purity of the ore; all of the reactions, however, serve to make the determination of the iron and silica more accurate and far more rapid.

The iron will soon be reduced to a protochloride, and a portion of the arsenic and antimony will be eliminated as arseniuretted and antimoniuretted hydrogen; the remainder of the antimony will be precipitated in the metallic state upon the zinc.

The lead sulphate is decomposed, forming metallic lead and zinc sulphate.

The chloride of silver is quickly decomposed into metallic silver and zinc chloride. Several other chemical reactions will ensue, but however, not pertinent to the subject-matter under consideration, as, for example, the deposition of metallic copper upon the zinc.

We have now lead, silver, antimony, and copper in the metallic state, mingled with the silica; and the iron in solution as a proto-salt. Filter, wash until filtrate shows the presence of no chlorides. The iron is estimated in the filtrate with potassium permanganate, after diluting largely (750 to 1,000 c.c.), and the addition of 20 c.c. concentrated sulphuric acid. The above precaution is necessary when manganese is present, or where the titration is performed in a hydrochloric acid solution. We are prepared to state that, by observing the above precautions, scientific accuracy may be attained.

Nitric acid (sp. gr. 1.2) is poured upon the filter residue, which readily dissolves the lead and silver, also the copper, if present. After removing the nitrates by hot water, the silica, with filter-paper, is transferred to a porcelain crucible, gently dried, then ignited as usual.

It will be seen that while we have detailed a method by which the lead and silver salts may be expeditiously removed from the silica, thereby greatly increasing the purity of the residue, yet we still have to contend with aluminium silicate and barium sulphate. We are free to confess, we know at present of no way in which these difficulties may be practically obviated. Fusion with sodium carbonate is wholly impracticable, when so many assays have to be made; the expulsion of the silica by means of hydrofluoric acid, in a platinum crucible, is perhaps the nearest approach we have for the solution of this very vexing problem.

In very many districts throughout the Western States, neither clay nor barite is found to contaminate the ores in an appreciable quantity; but the presence of much lead and silver are exceedingly liable to give analytical errors; in the obviation of which, we have found the above method invaluable.

M. W. Iles.

Grant Smelting Co., Leadville, Col.

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SCHOOL OF MINES NOTES.

ALEXANDER L. HOLLEY.

In the death of Mr. Holley, the School of Mines has lost the services of a man remarkable in many ways, possessing traits of character and accomplishments rarely united. A man of quick and accurate perception, of ready resource and remarkable ability, he added to genius most untiring industry. A student, and an original investigator in the field of science, a man of affairs, he was also an accomplished scholar, a brilliant speaker, and socially a most charming friend and companion.

Mr. Holley's eventful career has been described at length in the columns of the professional papers and magazines. A graduate of the scientific department of Brown University, his first work was in a railroad machine-shop, and afterward as a locomotive engineer, in which employments he laid the foundation of that practical knowledge of machinery, and the details of railway management, which contributed so much to his success in later years as an engineer and as an author of standard works on railroad practice.

Mr. Holley's name has been so intimately connected with the Bessemer process in this country, that in this place a mere mention of this, the work of the last eighteen years of his life, will suffice. One of his co-workers in this field, in a paper read before the Institute of Mining Engineers, says that it is "mainly through his inventions and modifications of the plant, that we in America are to-day enabled to stand at the head of the world in respect of amount of product."

Mr. Holley brought to his work in the School of Mines, in his lectures on the "Practical Metallurgy of Iron and Steel," the same thoroughness of preparation, logical arrangement of subject-matter, and clearness of presentation that distinguished his professional writings. Not content with the simple delivery of this most admirable course of lectures, he recognized at once the defects of the lecture system as a method of instruction, and spared neither trouble nor expense to increase their permanent value to the student. To insure that their notes should be as complete as possible, he prepared an outline or abstract, giving the salient points of each lecture, copies of which were furnished each member of the class. His lectures were illustrated with large diagrams copied from working drawings, and photo-litho-

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graphic copies of these diagrams, prepared at considerable personal expense, were also presented to each student. With these abstracts and illustrations the student could follow with profit Mr. Holley's lectures, in spite of his rapid delivery, and his concise, almost epigrammatic presentation of the subject, in sentences that admitted of no condensation.

From Dr. Raymond's eloquent tribute to the memory of his friend, we take the following sketch of the last year of Mr. Holley's life:

"A year has passed—for him, another year of zeal and activity, in spite of recurring weakness; a year in which, though he struck into no new line of inquiry, he performed some of the best professional work of his life, and perfected what may hereafter be recognized as his greatest invention. We refer to the movable converter-shell—an improvement in the Bessemer plant which bids fair to rank with the movable converter-bottom, and to carry to the utmost limit the productive capacity of the Bessemer process. It was while he lay desperately ill in London that he conceived the idea of this invention, and told a friend who called upon him, that he had 'been thinking, to pass away the time, and had thought of something which would enable' him 'to die with credit!' This humorous remark expressed a constant characteristic of his mind. His occupation as a consulting engineer, of detecting the weak points in methods and means, and suggesting the best remedies; of hunting up 'good things' and recommending them to his clients, was in exact consonance with his disposition. It enabled him to be constantly helping somebody.

"We cannot now lament that this year of fruitful activity on his part was not rather a year of rest. His impulse was wiser than the prudence of those who would have bid him forbear. For the autopsy performed since his death has shown that the malady under which he had already suffered, and which had puzzled the doctors, was incurable in its nature, and, we may fairly say, not directly traceable to overwork—a tumor obstructing the gall-duct, and producing many apparent symptoms of liver disease. He was attacked during a professional journey in Belgium, early in the autumn, with chills and fever, which he ascribed to the malaria of the low countries. In spite of the obstinate persistence of this trouble, he managed, before finally breaking down, to finish the work he had laid out in England and Scotland. Then he laid aside for the last time the pen and pencil that had done such splendid service—and the rest of the story is but a monotonous record of patient endurance, heroic courage and cheerfulness, unselfish thought for others, and a calm and absolutely unaffectedly fearless reception of the death-message."

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In the last number of THE QUARTERLY allusion was made to the "Report of theDirector of the Mint upon the Statistics of the Production of the Precious Metals in the United States." The full title was, however, not given, and it has been learned that those who followed our advice and wrote for the document received not this report but another. To such, we would advise another attempt, as both reports are worth having.

A REGULAR meeting of the Chemical Society was held on Thursday, January 12th, in Prof. Chandler's lecture-room, the president in the chair. After the regular routine business, a new method for the quantitative estimation of chromium in chromic iron ore was discussed by Mr. C. A. Wittmack ('82). The meeting then adjourned.

A SLIGHT change has recently been made in the method of reporting the results of the monthly examinations in the department of mathematics. Heretofore it has been the custom to post on the bulletin board the mark received by each student. Now, however, only the marks of those receiving below 6 are posted, leaving the others in a more or less pleasant state of uncertainty as to their vicinity to "6." This method renders a low mark doubly conspicuous, but it remains to be seen whether it will have any effect in raising the standard of scholarship. One of its effects will probably be to lend a feeling of security to students on the verge of final failure, which could easily have been avoided had they known their danger.

AT a recent meeting of the Faculty it was decided to drop from the roll the names of all students having unfulfilled conditions, and who could present no satisfactory excuse for such delinquency. To be thus summarily dropped is a calamity to any student, but it is felt with especial severity by those who have reached the fourth year, and have got, as has been aptly said, "within smell of Commencement." Nine of the fourth class students are in this dilemma, and to express their sincere sympathy and willingness to aid them the rest of the class promptly met and appointed a committee to draw up a petition for extra examinations, setting forth as strongly as possible the extenuating circumstances. Most of the conditions are in stoichiometry, and most of the conditioned are among those pursuing the chemical course. The petition was signed by the class and submitted to the Faculty. The arguments and extenuating circumstances detailed in the petition were acknowledged by that body, and a resolution was passed giving the delinquents another and a last chance to pass their examination the first week in March.

IN the matter of discipline it has not heretofore been the policy of the Faculty to administer correction by any of the usual milder forms of punishment, reprimand, suspension, etc. The theory has been that the students of a professional school, being engaged in the work of preparing themselves to earn their living as chemists, geologists, or engineers, will not wish or permit that work to be hindered or interrupted by disorder. Recent occurrences seem to indicate, however, that even the best and most earnest students need sometimes to be protected against themselves. Skylarking and practical buffoonery may

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be a good thing in a board of brokers, to relieve and divert the mind for the moment from the worry of business, but it is clearly out of place in a laboratory or other work-room, where the upsetting of a beaker, or interruption in the solution of a problem, may undo the work of hours.

THE NEW SCHOOL OF MINES BUILDING.—It has finally been recognized by the trustees of the College that the School of Mines has outgrown its present accommodations, and it has in consequence been decided to replace the present Fourth Avenue building by the erection of a new and larger building in its place. This decision has been hastened by a discovery, which was made in December, that the Fourth Avenue wall of the old building is several inches out of plumb. The great weight of the geological collections on the top floor, and of the mineralogical and metallurgical collections on the floor below, made it hazardous to allow the building to remain in this condition, and steps were at once taken to make the wall secure with heavy tie-rods reaching from wall to wall, and at the same time the floors were strengthened by additional posts on each story. It has now been decided to tear down this building as soon as the collections can be removed, and begin at once the erection of the new building. The wing, erected in 1874, at the south end of the Fourth Avenue building, and containing the lecture-rooms of the Geological and Chemical Departments and the Metallurgical laboratories, will not be torn down, but is to receive two additional stories. The old building between this wing and Fiftieth Street, built for the School in 1867, is to be rebuilt and made wider, and five stories in height instead of three. The basement, besides the store-rooms, will contain a large cloak-room, and an addition to the assay laboratory. The first floor will be occupied by the Museum of Applied Chemistry, and the second by the Department of Mineralogy and Metallurgy, as at present. The third floor will be occupied by the Department of Architecture, the fourth floor by the Engineering Department, and the top floor by the Geological Museum and lecture-room. The allotment of floors, above given, is, however, likely to be changed, the Department of Architecture taking the top floor and the Geological Department the third story. In consideration of the fact that three classes in the School of Mines, besides students from the college, attend Professor Newberry's lectures, while the students of architecture are, as yet, few in number, this latter arrangement will certainly suit the convenience of the greatest number.

The plans also contemplate a wing on Forty-ninth Street, occupying the site of the president's house, and extending from there to Fourth Avenue. It is, however, doubtful whether this wing will be built at present. The architect of the college, Mr. Haight, is now at work on the plans for the new building, which will probably be completed and submitted to the faculty and trustees for their approval within a few weeks. In such case we hope to be able to give our readers a more detailed description of the new building in our May number.

The history of the building about to be demolished is intimately connected with the early history of the school. During the first year

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of its existence, under Professors Egleston, Vinton, and Chandler, the school was lodged in two rooms in the basement of the old college building, the trustees having appropriated $500, and friends of the new undertaking $3,000, toward the fitting-up of a laboratory, with accommodations for twelve students. On the opening day, November 15, 1864, twenty-five men applied for admission, and before the end of the first year there were forty-eight in the school. The unexpected success of the undertaking, and the representations of President Barnard, who, from the beginning, gave the school most cordial endorsement and hearty co-operation, decided the trustees to assume the responsibility of providing for the financial support of the undertaking, not only through the current year, but also through the year ensuing —its further permanence to be dependent on the result. During the second year, therefore, appropriations were made for this school in the same manner as for other departments of the institution, and salaries were assigned to the professors.

The unexpected success of this first session encouraged the trustees to enlarge the accommodations of the school, which they did by appropriating $30,000 to fit up for its uses a building 70 by 30 feet in dimensions, on the college grounds, which had been previously used as a wall-paper factory, and providing room in the laboratory and drawing academy for seventy-two students. (This old paper factory was torn down in 1874 and replaced by the present Fiftieth Street School of Mines building.) The second year of the school, 1865–66, eighty-nine students were admitted, and the accommodations once more proved to be inadequate. At the close of the second year the experiment was pronounced a success, and other appropriations, amounting to $70,000 in all, were voted for the construction of a new building, the purchase of a complete equipment, and the estimated current expenses of the third year. Of this amount, $20,000 was expended in the building now about to be torn down, which, in view of the rapid growth, was made as large as could be built for the money, 120 feet long by 30 feet wide, or nearly double the size of the old paper factory.

These two buildings served till 1874, when again the school found its quarters too small, and the trustees found themselves obliged to provide additional accommodations. The demolition of the old paper-factory was therefore ordered, and the erection of the present Fiftieth Street building authorized. The cost of this building and its equipment was about $150,000.

BOOKS ADDED TO THE SCHOOL OF MINES LIBRARY SINCE DECEMBER, 1881.

MATHEMATICS AND ENGINEERING.

Colyer, Frederick. Hydraulic, Steam, and Hand-Power Lifting and Pressing Machinery. 8vo, 120 pp., 73 drawings. London, 1881.

City of New York *vs.* Tilly Foster Iron Mine. Supreme Court. Application of the Mayor, Aldermen, and Commonalty of the City of New York to take Lands in Putnam County for a Storage Reservoir. 8vo. 1875.

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Gorsuch, Robert B. The Mexican Southern Railway. 8vo, 25 pp. New York, 1881.
Lavoisine, E., and E. Pontzen. Les Chemins de Fer en Amérique. 8vo, and Atlas, 4to. Paris, 1880.
Nexsen, Heyer M. Short-Hand Arithmetic. 12mo. New York.
Notes on Building Construction. Arranged by the Committee of Council on Education, South Kensington. 3 vols. 8vo. London, 1875.
Reports on Pumping-Engines. By Wm. E. Worthen, C.E., and others. Water-Works of the Cities—Brooklyn, Hartford, Belleville, Cambridge, Lynn, Milwaukee, Lawrence, and Long Island City. 1857 to 1877.
Reports to the New York Commissioners of Docks. By Gen. John Newton, Gen. Q. A. Gillmore, and Wm. E. Worthen, C.E. 8vo. New York, 1876 and 1881.
Schlockow, Dr. Die Gesundheitspflege und medicinische Statistik beim Preussischen Bergbau. 8vo, 310 pp. Berlin, 1881.
Waring, George E. The Sewerage and Drainage of Cities. Five Pamphlets. 1879–1881.

CHEMISTRY AND PHYSICS.

Bertholet, M. Essai de Mécanique Chimique, fondée sur la Thermochimie. 2 vols. 8vo. Paris, 1879.
Elliott, Arthur H. An Apparatus for the Rapid Analysis of Mixtures of Gases. 8vo, 8pp. New York, 1881.
Gamgee, Arthur. A Text-Book of the Physiological Chemistry of the Animal Body. Vol. I., 8vo, pp. 487. London, 1880.
Remsen, Ira. Report on the Impurity of the Boston Water-Supply. 8vo, 23 pp. Boston, 1881.
Wislicenus, Johannes. Adolph Strecker's Short Text-Book of Organic Chemistry, translated by Hodgkinson and Greenaway. 8vo, 789 pp. London, 1881.

GEOLOGY AND NATURAL HISTORY.

Dana, James D. Geological Relations of the Limestone Belts of Westchester County, New York. 8vo, 119 pp. New Haven, 1881.
Newberry, J. S. The Genesis and Distribution of Gold. 8vo, 14 pp. New York, 1881.
Paetel, Fr. Molluscorum Systema et Catalogus. 8vo, 119 pp. Dresden, 1869.

JOURNALS AND TRANSACTIONS (NEW).

Institution of Mechanical Engineers. 34 vols. 8vo, and index vol. 1847–1881.
Science Observer. Vol. 3 (Serial). Boston, 1881.
The Observatory. A Monthly Review of Astronomy. Vols. 1 to 4. London, 1878–1881.

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GRADUATE DEPARTMENT.

THE ALUMNI DINNER.—The Twelfth Annual Reunion of the Alumni Association of the School of Mines took place at the Westminster Hotel, on December 29th. It was one of the largest and most successful meetings ever held, and was thoroughly enjoyed by all present. Having assembled in the parlors of the hotel, the members afterward adjourned to the dining-room, where they did justice to the many delicacies set before them. The meeting having been called to order by the President for the year 1882, J. P. Carson, '68, resolutions, deploring the untimely death of our fellow-graduate and late President, Archibald MacMartin, were adopted on motion of W. A. Smith, '68, who gave an account of the short and useful life of the deceased, and the many sterling qualities which endeared him to all who knew him. Upon motion of P. de P. Ricketts, '71, a committee of three were appointed to draft resolutions commemorative of the members of the Alumni who have died during the year: Archibald MacMartin, '68, H. G. Clark, '77, and O. F. Olmsted, '78.

The first regular toast of the evening, The "School of Mines Quarterly," was answered by H. S. Munroe, '68, who described in concise and appropriate language the scope and aims of the QUARTERLY. This toast was followed by a vivid description of "The Coast Range, and What I Know about Beans," by B. B. Lawrence, '78. Some "points" on "astronomy" were elucidated by the Director of the new Observatory, J. K. Rees, '75. P. de P. Ricketts, '71, spoke at length in response to the toast, "The Alumni Association," referring to the present reunion and the work and future of the Association. Mention was also made of the large number of members present.

Quiet having been restored in the "laboratory" by the efforts of A. L. Colby, '81; A. D. Churchill answered to the toast of '80, and W. H. Van Arsdale, '68, gave a detailed account of "Rocks and Chinamen," aided by the President. A. H. Elliott spoke at great length on "Gas Analysis, Petroleum, and the Class of '81," K. Neftel answered for the Class of '79, F. G. Wiechmann, '81, made a few impromptu remarks in response to the toast to his class.

Elwyn Waller, '70, spoke on "Chemistry" in great detail. "The Class of '79 and the Study of Geology" were commented upon by N. L. Britton. Next followed speeches by E. M. Parrot, '70; J. B. Mackintosh, '77; A. L. Beebe, '80, and several others, on the "Quarterly" and various other topics of interest. The resident secretary, P. de P. Ricketts, '71, then announced that he had received a great many letters, only a few of which could be read, owing to the advanced hour.

It was greatly regretted by all present that Prof. Trowbridge, the

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invited guest of the evening, was absent. The following is a list of the officers for 1882:

President, J. P. CARSON, '68.
First Vice-President, WM. H. VAN ARSDALE, '69.
Second Vice-President, J. K. REES, '75.
Third Vice-President, K. NEFTEL, '79.
Fourth Vice-President, A. D. CHURCHILL, '80.
Fifth Vice-President, A. L. COLBY, '81.
Treasurer, W. E. HILDRETH, '77.
Secretary, H. S. MUNROE, '68.
Res. Secretary, P. DE P. RICKETTS, '71.

C. A. W.

MEMORIAL RESOLUTIONS.

THE Alumni Association of the School of Mines of Columbia College, wishing to place on permanent record their appreciation of the worth of three of their number who have been removed by death during the year 1881, and their sense of bereavement in the loss which they have sustained, do adopt the following resolutions:

Whereas, It has pleased the Almighty Father of us all to remove from us by death, during the past year:

ARCHIBALD MACMARTIN, of the Class of 1868; HENRY GILBERT CLARK, of the Class of 1877, and OWEN FREDERICK OLMSTED, of the Class of 1878.

Resolved, That we, as an association, while recognizing in all the events of life the hand of an all-wise Providence, do as men deeply deplore our loss, and have heartfelt regret in the thought that we shall never see again, in our midst, the familiar faces of these our former associates.

Resolved, That in the record which each one of these, our associates, has left behind him—both as a student and as a man—we see cause to congratulate ourselves in having been so closely associated with men of high character, earnest purpose, and successful achievement.

Resolved, That in the memory of those who have left us, our Association has a precious treasure, which should serve to us, and those who come after us, as an example of industry and loyalty, and as an incentive to labor and usefulness.

Resolved, That we respectfully tender to the relatives and friends of the deceased the expression of our cordial sympathy in the more personal loss which has been theirs.

Resolved, That these resolutions be published in the next number of the SCHOOL OF MINES QUARTERLY.

WILLIAM ALLEN SMITH, '68,
CHARLES LOUIS CONSTANT, '77,
CHARLES EDWARD MUNSELL, '78,
} *Committee.*

NEW YORK, January 11, 1882.

FOR lack of space we are compelled this month to omit the usual extracts from our graduate correspondence. The following list, however, gives the address and occupation of each graduate of the School so far as known. This list will be published again in the May number of the QUARTERLY, and it is to be hoped that all concerned will send us word as soon as possible of changes of address or other corrections which should be made.

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LIST OF GRADUATES.

1867.

John Magnus Adams, A.B., A.M., E.M., Mining and Mechanical Engineer, Agent for Frue Vanner, etc., 109 California Street, Room 7, San Francisco, Cal.

Samuel Willard Bridgham, E.M., 49 West Twenty-third Street, New York City.

Francis Gordon Brown, E.M., Merchant, 116 East Twenty-seventh Street, New York City.

Edward Stelle Bronson, A.B., A.M., E.M., Prospect Park, Brooklyn. House, 144 Hicks Street.

John Adams Church, E.M., Ph.D., Superintendent Tombstone Mill and Mining Co., Tombstone, Arizona.

Henry Bedinger Cornwall, A.B., A.M., E.M., Professor of Analytical Chemistry and Mineralogy, College of New Jersey, Princeton, N. J.

Edward Everett Giddings, E.M., Merchant, 21 Metropolitan Block, Chicago, Ill.

Charles King Gracie, A.B., E.M., Consulting Engineer, with J. A. Church, 115 Boreel Building, Broadway, New York City.

Albert Ward Hale, A.B., A.M., E.M., Civil and Mining Engineer, 71 Broadway, New York City.

Thomas Hayes Harmer, A.B., A.M., E.M., 113 East Thirty-seventh Street, New York City.

Frederic Milton Heath (formerly Petit), E.M., Potsdam, Saint Lawrence County, N. Y.

William Wey Tuttle, E.M., Horticulturist, care of Baird & Tuttle, Bloomington, Ill.

David Van Lennep, E.M., County Surveyor and Assayer, Winnemucca, Humboldt County, Nevada.

1868.

William Henry Van Arsdale, A.B., A.M., E.M., Consulting Engineer, 16 Exchange Place, New York City.

Augustus Porter Barnard, E.M., R. R. Engineer, Sanford, Orange County, Florida. Address, care of D. Van Nostrand, 23 Murray Street, New York City.

George Strong Baxter, A.B., E.M., Mining and Civil Engineer, Cashier, North Pacific Railroad, 23 Fifth Avenue, New York City.

James Pettigrew Carson, E.M., Mining Engineer; address, 82 Broadway, New York City.

Albert Huntington Chester, A.M., E.M., Ph.D., Childs Professor of Agricultural Chemistry, Hamilton College, Clinton, Oneida County, N. Y.

George Hampton Coursen, E.M., C.E., 307 Greenwich Street, New York City.

George Jarvis Geer, Jr., E.M., Merchant, 453 Broome Street, New York City.

George Byron Hanna, A.B., E.M., Melter, U. S. Assay Office, Charlotte, N.C.

*Archibald MacMartin, A.B., A.M., E.M., Editor and Metallurgist, 168 Fifth Avenue, New York City. Died in New York City, May 7, 1881.

Edward Stewart Moffatt, A.B., A.M., E.M., Superintendent of the Musconetcong Iron Works, Stanhope, N. J.

George Howland Parsons, E.M., Superintendent of Land Improvement Company, Colorado Springs, Col.

William Pistor, E.M., Hamilton & Pistor, Architects, 35 Broad Street, New York City.

Charles Slason Platt, E.M., Assayer, 4 Liberty Place, New York City.

Kenneth Robertson, E.M., Superintendent of the Missouri Furnace Company. Address, care of Potter & Riggs, St. Louis, Mo.

Albert P. Shack, E.M., Ministry, 24 Somerset Street, Boston, Mass.

Frederick Augustus Schermerhorn, E.M., Trustee of Columbia College. Address, 61 University Place, New York City.

Lenox Smith, A.B., E.M., Steel Rails, 46 Pine Street, New York City.

William Allen Smith, E.M., Manager of the Weldon Mining Company, Dover, N. J. Address, 16 Exchange Place, New York City.

Frederick Stallknecht, E.M., Editor, 133 West Thirteenth Street, New York City.

Moses Dillon Wheeler, A.B., E.M., P. O. Box 231, Stapleton, Richmond County, Staten Island.

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1869.

* Thomas Monahan Blossom, A.B., A.M., E.M., died, October, 1876, in California.
Frederick Bruckman, E.M., Assayer to the Amie Consolidation Company, Leadville, Col.
Alonzo Clarence Campbell, E.M., Chemist and Mining Engineer, 227 Shelby Avenue, Nashville, Tenn.
William Augustus Hooker, A.B., A.M., E.M., Consulting Geologist and Mining Engineer; office, 2 Wall Street, New York City.
Roland Duer Irving, A.M., E.M., Ph.D., Professor of Geology and Mineralogy in the University of Wisconsin, Assistant State Geologist of Wisconsin, and Special Agent of the Tenth Census of Exploration of Lake Superior.
Walter Proctor Jenney, E.M., Ph.D., Consulting Engineer and Geologist, Deadwood, Dakota.
Henry Smith Munroe, E.M., Ph.D., Adjunct Professor in Surveying and Practical Mining, School of Mines, Columbia College.
Lionel Robert Nettre, E.M., Superintendent of the Gregory Consolidated Mining Company, Helena, Lewis and Clarke Counties, Mont.
* Henry Newton, A.B., E.M., Ph.D. Died, August 5, 1877, at Deadwood City, Dak. Terr., while occupied as Assistant Geologist of Black Hills Expedition.
William Bleecker Potter, A.B., A.M., E.M., Professor of Mining and Metallurgy, Washington University, St. Louis, Mo.
John Cooper F. Randolph, A.B., A.M., E.M., Consulting Engineer, 35 Broadway, New York City.

1870.

Ogden Haight, E.M., Stock Broker, 38 Broadway, New York City.
William Halsey Ingersoll, A.B., A.M., LL.B., E.M., Portland, Conn.
John Augustus Knapp, A.B., A.M., E.M., General Manager Mingo Furnace, P. O. Box 515, Salt Lake City, Utah.
John Leo Lilienthal, E.M., P. O. Box 1058, San Francisco, Cal.
Stuart Lindsley, E.M., Superintendent, Detroit Copper Company, Clifton, Ariz.
Edward Moore Parrot, E.M., Manager of the Furnaceville Iron Company, Furnaceville, Wayne County, N. Y.
Richard Henry Terhune, E.M., Superintendent of the Morgan Silver Smelting Company, Salt Lake, Utah.
Theodore Francis Van Wagenen, E.M., General Manager Twin Lakes Consolidated Gold Placer Mining Company, Leadville, Col.
Elwyn Waller, A.B., A.M., E.M., Ph.D., Instructor in Analytical Chemistry, School of Mines, Columbia College, and Chemist to the New York Board of Health, New York City.

1871.

William E. S. Fales, E.M., LL.B., Lawyer, 359 Fulton Street, Brooklyn, N. Y.
Samuel Anthony Goldschmidt, A.B., E.M., Ph.D., Manufacturing Chemist, Columbia Chemical Works, 66 Water Street, Brooklyn, N. Y. House, 709 Madison Avenue, New York City.
John Gordon, Jr., E.M. Address, care E. Johnson & Co., Rio Janeiro, Brazil.
Pierre de Peyster Ricketts, E.M., Ph.D., Instructor in Assaying, School of Mines, Columbia College.
George Washington Riggs, Ph.B., Chemist, Potter & Riggs, Washington University, St. Louis, Mo.
Gracie Sayre Roberts, E M., C.E., Consulting Civil Engineer, 88 Union Avenue, Brooklyn, E. D.
Richard Spotswood Robertson, Jr., E.M., with Cargenie Bros. & Co., Pittsburg, Pa.

1872.

Peter Townsend Austen, Ph.B., Ph.D., F.C.S., Professor of Analytical and Applied Chemistry, Rutgers College, New Brunswick, N. J.
* Frank B. Jenney, E.M., died in Trinidad, 1876, while engaged as Superintendent of the Orinoco Exploring Company.
Frederick H. McDowell, E.M., Contractor and Engineer, 17 Cortlandt Street, New York City.

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Thomas O'Connor Sloane, A.B., A.M., E.M., Ph.D., Chemist, Sloane & Rupp, 119 Pearl Street, New York City.
Arthur F. Wendt, E.M., C.E., Consulting Engineer, 414 East Fifty-first Street, New York City.

1873.

Frederick A. Canfield, A.B., A.M., E.M., Consulting Engineer, Dover, N. J.
Charles Adams Colton, E.M., Assistant in Mineralogy, School of Mines, Columbia College.
Henry Augustus Mott, Jr., E.M., Ph.D., Mining Engineer and Analytical Chemist; office, 105 Water Street; residence, 30 West Fifty-ninth Street.
Henry Walter Webb, E.M., Lawyer, Webb & Sprague, 206 Broadway, New York City.
John Townsend Williams, E.M., Ph.B., East River Chemical Works, Forty-fourth Street and East River, New York City.

1874.

Charles Sumner Allen, Ph.B., M.D., Practising Physician, 104 East Eighty-fifth Street, New York City.
William de Liesseline Benedict, E.M., Consulting Mining Engineer, with Union Ore Conc. Co., 29 Broad Street, New York City.
John Gedney Mott Cameron, E.M., C.E., Assayer, Spears & Howard; address, Fifty-fifth Street and Madison Avenue, New York City.
Samuel Morris Lillie, E.M., Chemist, Harrison, Havemeyer & Co., 101 South Front Street, Philadelphia; residence, 307 Pine Street, Philadelphia.
George Murray, E.M., 235 West Twenty-third Street, New York City.
Eben Erskine Olcott, E.M., Superintendent St. Helena Gold Mine, Delicias, Sonora, Mex. Address, 111 West Thirteenth Street, New York City.
Benjamin Franklin Rees, E.M., Merchant, 37 Ferry Street, New York City.
Francis Bell Forsyth Rhodes, E.M., Superintendent Mingo Works, near Salt Lake, Utah.
Frederick Harrison Williams, E.M. Care of W. Williams, Orange, N. J.

1875.

Magnus, C. Ihlseng, E.M., C.E., Ph.D., Prof. of Engineering, State School of Mines, Golden, Col.
Malvern Wells Iles, Ph.D., Assayer, Grant Smelting Works, Leadville, Col.
Charles Edward Jackson, C.E., Division Engineer, Manhattan Railroad Company, 60 Gold Street, New York City.
Douglas Arad Joy, E.M., Assistant in Chemistry, University of Michigan, Ann Harbor, Mich.
*Robert Schuyler Lamson, C.E., Major in the Egyptian Army, died at Darbour, Upper Egypt, 1876.
Harry Wenman Leavens, E.M., U. S. Geological Survey, San Francisco, Cal., also care of Frederick Leavens, 835 Broadway, New York City.
Arthur Macy, Ph.B., C.E., Supt. Silver Mt. Mining Co., Ophir, Col. Address, 35 Broadway, New York City.
William Skaats Noyes, E.M., Assayer in charge of Bodie Mill, Cal.
Philip Charles Pfister, E.M., 125½ Forsyth Street, New York City.
Franklin Pool, E.M., Chemist, Celluloid Manufacturing Company, Newark, N. J.
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James Simpson Chester Wells, Ph.D., Instructor in Analytical Chemistry, School of Mines, Columbia College.
Edwin Atwater Wetmore, E.M. Care of E. A. Wetmore & Co., Iron Merchants, Marquette, Mich.

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1876.

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Frederick Everett Bruen, E.M., C.E., Supt. Gallespie & Ballou Sampling Works, Leadville, Col.
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Elbert Champlin Van Blarcom, C.E., Tombstone, Ariz.
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1877.

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Augustus Cass Canfield, E.M., 60 West Fifty-fourth Street, New York City.
John Britton Cauldwell, C.E., 6 East Forty-ninth Street, New York City.
*Henry Gilbert Clark, E.M., C.E., Late Supt. Cortez Mining and Milling Co., Aurora, Nev. Died of typhoid fever, August 2, 1881.

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William Henshaw Radford, E.M., Richmond Mine, Eureka, Nev.
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1878.

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1879.

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Harry Clay Cornwall, E.M., Assayer, Irwin, Gunnison Co., Col.
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1880.

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1881.

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ENUMERATION.

Class of 1867	13	Class of 1876	26
" 1868	20	" 1877	31
" 1869	11	" 1878	40
" 1870	9	" 1879	33
" 1871	7	" 1880	36
" 1872	5	" 1881	36
" 1873	5		
" 1874	9	Total number of graduates	301
" 1875	20		

Known to be deceased.

Class of 1868	Archibald MacMartin.
" 1869	Thos. M. Blossom.
" 1869	Henry Newton.
" 1872	F. B. Jenney.
" 1875	Robert S. Lamson.
" 1877	Henry G. Clark.
" 1877	James R. Priest.
" 1878	Owen F. Olmsted.
" 1879	Francis B. Stewart.
Total	9

Number of graduates living 292

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Employed as follows:

As Mining Engineers, consulting	22	
" " " field-work	10	
" " " superintendents of mines	16	
" " " instruction	4	52
" Civil and Mechanical Engineers, consulting	2	
" " " " field-work	33	
" " " " instruction	6	
" " " " sanitary work	3	44
" Chemists, consulting and analytical	38	
" " superintendents of works	3	
" Assayers	31	
" Chemists and Assayers, instruction	15	87
" Metallurgists, consulting	7	
" " superintendents of works	12	19
" Geologists and Mineralogists, consulting	2	
" " " " field-work	4	
" " " " instruction	6	12
" Architects		4
" Lawyers		5
" Artists		1
" Physicians		1
" Editors		1
" Ministers		1
In Agriculture		2
" Business		29
" Manufacturing		8
" Studying, Europe and elsewhere		14
" Travelling		2
Occupation unknown		10
Total		292

	Per cent.	
Employed as Mining Engineers	17.27	
" " Civil and Mechanical Engineers	14.61	
" " Chemists and Assayers	28.90	
" " Metallurgists	6.32	
" " Geologists and Mineralogists	3.99	
" " Architects, Lawyers, etc	4.99	
" in Professional work, Total		76.08
" " Business and Manufacturing		12.29
" " Studying and Travelling		5.32
Occupations unknown		3 32
Deceased		2 99
		100.00

Degrees Conferred to Date.

E.M	198
C.E	79
Ph.B	63
Ph.D	27
Total	367
Deduct for repetitions	66
	301

In the classification of the graduates as to employment, the principal occupation only of each man is taken into consideration. Many of the mining engineers act as chemists, geologists, etc., and vice versa. The list is correct to February 1, 1882.

P. DE P. RICKETTS,
Resident Secretary Alumni Association.

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THE

SCHOOL OF MINES QUARTERLY.

VOL. III. MAY, 1882. No. 4.

THE MARQUETTE IRON REGION.—IV.

BY THE STUDENTS[1] OF THE SUMMER SCHOOL OF PRACTICAL MINING, LAKE SUPERIOR, 1881.

HOISTING.

THE ore is hoisted to the surface in buckets, skips, or cars. Buckets are used only in open cuts, small temporary workings, and in sinking winzes, etc.; skips are used in hoisting from all the underground workings, except at shaft 5 and 6 of the Republic Mine, where the mine cars are run on to a cage and hoisted with the ore they contain. The cages are built of plate and angle iron, and weigh about three tons. They are provided with safety-clutches, so constructed that should the cable give way, two cam-shaped segments, provided with shallow teeth, would be made to seize the guide-rods and thus prevent a fall of the cage. In one instance the rope broke while a man was in the cage, but the cage only dropped a few feet before the safety-clutches came into play and stopped its further descent. It is stated that the man was unaware of the accident, and upon the cage stopping, called out to be hoisted higher.

Various methods of signalling are employed for communi-

[1] Being extracts from the memoirs required of each student of the Summer School.

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cation between the engine-house and the mine. Generally the apparatus consists simply of a wooden lever, weighted to render it less sensitive, and placed in the shaft-house. The lever is connected with the mine so that when a wire is pulled and suddenly released the lever is raised and allowed to fall, striking against an iron plate. The signals are usually sent from the shaft-house to the engine-room by bells. At the Jackson Mine electricity is used entirely, and a very elaborate code of signals has been employed for some time.

The position of the skip is generally indicated in the engine-room by means of "tell-tales." These are sometimes of elaborate construction, worked by screw-gearing from the drum-shaft, or more simply by cords winding on a small reel; but in some cases the "tell-tale" consists of a piece of hemp tied to the hoisting-rope.

As a rule the hoisting-houses are at a considerable distance from the shaft mouths, in some convenient and central location. At the Jackson Mine each drum is run by a special engine, and small hoisting-houses are set up where needed; but this is exceptional, the general plan being to concentrate in one or two buildings the hoisting-drums for all the shafts, and to couple several drums to one engine. At the New York Mine, where an old and simple type of apparatus is used, there are four drums run by one engine, as follows: Two parallel shafts are in gear with the driving-shaft, and are in motion whenever the engine is; two drums turn loosely on each shaft, but may be thrown into gear by ordinary clutch-coupling worked by levers. The levers have toothed arcs at their extremities, and are moved by small pinion-wheels turned by horizontal hand-wheels.

The following is a description of one of the hoisting-drums used at the Republic Mine: The drum is made of cast-iron, the ends and face-plates being cast separately and bolted together. The ends are wheel-shaped six-armed pieces, having an inner flange to which the face-plates are bolted. The face-plates are bolted to the end-pieces and to each other by means of flanges on their interior surfaces, and are so cast that when fitted together the grooves on their surfaces form a perfect helix.

Some of the newer hoisting plants are being provided with internal gearing, the working of which may be described as follows: On the shaft is fixed a small spur-wheel, which is

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in gear with two others of the same size, the axes of which are attached to a brake-wheel of the diameter of the drum, and, like it, loose on the shaft, and these in turn mesh into a rack on the inner surface of the drum. When the loose brake-wheel is free to move, it simply revolves on the shaft, but when the brake is applied to it, the small wheels, as they revolve, take hold of the drum (by the inside rack) and cause it to revolve. This form of gearing allows of regulating the speed of the hoist without altering the speed of the engine, and has been quite successful, especially for small hoisting plants.

The ordinary form of friction gear, with V-grooves on drum and driving-pinion, is in use at most of the mines. The disadvantage, however, of this so-called "V-friction" is due to the tendency of the pinion to produce slight indentations in the grooves of the larger friction-wheel whenever thrown into gear. These increase in number and size, and finally give rise to an irregular curve which prevents the proper working of the drums.

At the Lake Superior Mine the Lane drum is used. In this the driving power is applied directly to the drum-shaft, which communicates it to the drum by a band-clutch. A band-brake is so arranged with the band-clutch that both may be operated by a single hand-wheel, or each by a separate hand-wheel. A differential device is provided for taking up the wear of the driving and brake bands; also a device for holding the band-clutch from contact with its pulley when not turning the drum.

The Republic Mine effects a great saving by utilizing the water-power of the Michigamme River for hoisting, pumping and drilling purposes. A short distance below the mine, at a succession of rapids in the river, a canal has been cut, about half a mile long, 30 feet wide and 8 feet deep, near the centre of which are placed three guard-gates with an equal number of head-gates, through which the water is supplied to the wheels. At the lower end of the canal is the building, in which there are four air-compressors, 24×60 inches. These are driven by two turbine-wheels, 66 inches in diameter, with 16 feet fall of water. Each turbine is provided with a separate and distinct flume; and each flume with a grating for removing anything likely to clog the wheels.

The vertical shafts of the turbines are terminated on the upper end by bevel wheels, gearing into vertically moving wheels fixed to horizontal shafts. These shafts by suitable

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gearing drive parallel shafts which work the compressors. Each turbine drives two double-acting compressors.

The air from all the compressors goes to a receiver, 48 feet 4 inches long and 4 feet in diameter, placed underneath the building. From this receiver the air is conveyed to the different plants at the mine through a wrought-iron pipe, 16 inches external diameter and 4,300 feet long, of ¼-inch boiler-iron, made in lengths of about 50 feet and fastened together by flanges 1¼ inch thick. At intervals of 500 feet there are brass-lined expansion joints, permitting a movement of 12 inches, and at short intervals gas-pipe rollers, resting on iron strips, allow the pipe to move easily when expanding or contracting.

The supply of water for running the turbines can be had for eight months every year. During the remainder of the time the compressors are run by two horizontal steam-engines, 22 × 28 inches.

The four compressors combined furnish enough power to drive 20 drills, 8 hoisting-engines, 17 water-pumps. Calculation from measurements of the compressors shows that, all together, they are capable of supplying 2,136 cubic feet of air per minute at a pressure of 40 pounds per square inch.

PUMPING.

Whenever the quantity of water is large or the depth great, Cornish plunger and suction pumps are used. For comparatively shallow workings, or for workings not directly connected with the pump-shaft, steam-pumps are employed. At the New York Mine and at the Cleveland, where Cornish pumps are used, the connections are so arranged by bobs and wire ropes that the main rod is balanced by the others. In one case the rod in the main pump-shaft is connected by a wire rope, passing through a drift 85 feet long, to a rod working in a vertical winze.

At the Michigamme Mine the pumps are worked by wire ropes instead of rods, directions being changed and ropes balanced by bobs in the usual way. At intervals of 50 feet the ropes are supported on sheaves and rollers. The water-pipe is of wood, coated heavily with a mixture of sawdust and tar. A portion of the water is conducted into reservoirs to be used in washing the ore on the picking-tables.

On account of the comparative shallowness of the Jackson

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Mine the pumping arrangements are very simple. Wherever water is found to be troublesome, a Blake or Knowles pump is placed at the bottom and pipes for steam and water run to it, the water being forced directly out in one lift. The steam-pipes are lagged in the ordinary way, and in addition, when much exposed, are surrounded by common tin or iron drain-pipe. This is loose and open at the top, and through it the exhaust steam is allowed to ascend, keeping the steam-pipe warm and reducing the loss from condensation.

CONTRACT WORK.

A considerable portion of the mining is done by contract work. In some cases one man contracts to get out all the ore, and sublets to smaller contractors. In other cases each pit is contracted for separately, and at one mine even the blacksmith work is let out on contract. In still other cases, and at most of the mines, separate contracts are made with small parties of men for stoping, filling, drifting, etc., while the surface work is usually company work.

At one of the mines very close contracts are made—so close that it is usually necessary to guarantee the contractor a certain sum per month, about the wages of a pit boss, and this sum is not very often exceeded. As a further precaution against an unexpectedly advantageous result to the contractor, the agreements are so worded that they can be annulled at the will of the superintendent.

In a recent contract at this mine a peculiar plan has been adopted. A strict detailed itemized account is kept with the pit, and an expense statement is made up of:

Amount paid to hand-drillers.
" to fillers.
" to blaster and labor.
" for candles and oil.
" for explosives.
" for air for machines.
" for repairs of machines.
" for oil for machines.
" for labor for machines.
" drill carrier.
" for other repairs.

Allowance to the contractor $80 per month.

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The cost thus made up is credited by the ore raised. Then if at end of time worked the cost of the ore is found to be on an average

$1.35,	allow an increase of	10	per cent.	to	salary, or	$88	per month.
1.25,	"	" 20	"	"		96	"
1.20,	"	" 25	"	"		100	"
1.15,	"	" 35	"	"		108	"
1.10,	"	" 40	"	"		112	"
1.00,	"	" 50	"	"		120	"

Whenever the company guarantee a contractor a certain salary, they usually also set a limit to the amount which the man shall make—anything beyond this going to the company. For example, "If said Fossberg and Bystrom do not make two dollars and fifty cents per day by the contract, the company will pay them that amount, and if they make over one hundred dollars per month each, the balance shall go to the company."

In other cases the company becomes an equal partner in all over a certain sum per month. In another case, "If the contractors should not make enough to pay advances made by the company, and should work at any future time in the mine, any excess which they might make over seventy dollars per month shall be retained till all such advances be paid off."

At another mine the contract to mine all the ore, in the two largest pits, is given to one man at so much per ton sent to the surface, he furnishing light, tools, powder, etc., and doing all the work; the company doing the timbering, hoisting, and pumping. The company guarantees that the workmen shall receive a certain sum per month. The contractor in turn lets out part of the work to other contractors, but has most of it done by men who are paid according to the amount of work they do—so much a ton or car-load for the ore they shovel, or so much per foot for the drilling.

Prices paid are:

Hand drilling.....80c. to $1.50 per foot.
Machine drilling.....50c. to 60c. "

A man taking a contract buys from the company what tools he may need; on finishing the contract the tools are examined, and if in good order, the contractor is paid back the amount he gave for them, less forty-seven cents per pound for steel he has used up.

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The contractors pay the company $80 per cwt. for glukodine, which is worth $37 only, though it is very likely that it would pay the company better to sell the powder at cost price, as much more ore would be sent up with the same amount of work. As it is, much time is wasted in sledging to save a little powder.

PAYMENT.

The wages paid at one of the mines are:

Common labor	$1.45	Teamster	$1.50
Timber-men	1.70	Ore-weighers	1.60
Trestle-men	1.50	Engineers	1.75–2.00
Machine-men	2.00–2.25	Firemen	1.65–1.70
Helpers	1.80	Mechanics	2.50–3.00
Boss timber-man	2.50	Diamond-setters	3.00
Pit bosses	2.50		

The prices paid at another mine are 80 cents to $1.50 per foot for hand drilling, 50 to 60 cents per foot for machine drilling. Three men will drill 8 to 12 feet per day.

Saturday is not generally selected as pay-day: if a certain date is selected, and this falls on Saturday, payment is often deferred until the following Monday.

The system of payment at the Michigamme Mine works very smoothly. Its chief peculiarity is hinted at in the following clause taken from the "Rules and Regulations:" "Any accounts not examined on examination day will not be corrected." On the first of each month the amount of the previous month's contracts are made out, then copies of these contracts are sent to the spokesman of each contracting party. The third day of the month is set apart by the captain, timekeepers and clerks as examination day, and any contractor on that day has a right to bring his contract for correction and revision if necessary. If there is any error in the accounts, this is the day on which they must be corrected. Two days before pay-day, the third Saturday in the month, detailed statements of the amount due each man are distributed to the men, and at the foot of each is a receipt. The men have these two days for the correction of any error in their individual accounts. Each man as he presents, on pay-day, the signed statement, receives in return an envelope on which is his name, the number of his check, and the amount due him.

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ACCIDENTS.

At one of the oldest mines, and during the last thirty-three years, but fourteen men have been killed, as follows: two men caught in the tunnel by the locomotive; powder-man killed by the explosion of the magazine; two or three men by falls in the stopes; and the rest by getting under rocks which they were prying off. The common accidents to blasters, such as burnt faces, blindness, and mutilated fingers, occur with considerable frequency.

At another mine thirteen men have been killed in the last eighteen years: six by falling rock; one by the explosion of nitro-glycerine; four by caving in of a pit; one fell into a pit; one unknown.

By far the larger number of accidents at a third mine have resulted from the explosives. The first was the total demolition of the powder-house and of the two employés in charge. The next was the explosion of a can of nitro-glycerine while being carried. The man who was carrying it is supposed to have stumbled and fallen. He was killed. The third was caused by striking with a machine drill some nitro-glycerine which had lodged in a crevice. The man who was pouring water into the hole was killed.

It has been ascertained that nitro-glycerine instead of exploding at once will sometimes boil in the hole for a long time and may finally explode. In one case the men returned to examine the hole after fifty-two minutes had elapsed, and found the nitro-glycerine in a violent state of ebullition. They ran away again, and in about five minutes the charge went off.

The careless thawing out of frozen nitro-glycerine gives rise to many accidents, of which the following, from the Champion Mine, may be taken as a type: Two miners were employed in thawing out nitro-glycerine in a pail of water, and in order to get the water warmer they threw in a piece of hot iron. As neither the men nor the pail have since been found, it is probable that some nitro-glycerine had leaked out from some former can and was exploded by the impact of the iron. The firing of block-holes is also a prolific cause of accident. The following occurred at the Champion Mine: One of the men had charged a block-hole, lighted the fuse, and retired; after waiting some time he heard what he supposed was his blast, and returned just as the blast exploded. He was blinded and narrowly escaped death.

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RELIEF FOR THE INJURED.

At the Jackson Mine the doctor is secured a salary of nearly $2,000 per year, and his name entered on the pay-roll as one of the regular employés. The men are charged for this; those with families paying more and boys paying less than single men, but no one employed is exempt from their regular charge. The doctor is allowed any outside practice that will not interfere with his care of the miners. The Jackson Relief Club was entirely separate from the mine until one day the funds and treasurer were missing. Since then the mine paymaster has acted as their treasurer, and the company pays them interest on all that it holds. The members pay a regular due, and when disabled receive a weekly allowance.

Two serious accidents at the Michigamme Mine in the early part of last year, from the premature explosion of glukodine, induced the miners to start a fund for the relief of sufferers in case of future accidents. This movement is to be liberally supported by the company. The scheme as at present understood is as follows: Each man is to contribute 30c. per month, and for every $100 contributed by the men the company is to add another $100. This is to continue till the fund amounts to $10,000, at which figure it is proposed to maintain it. In case of death from injury the deceased man's family is to receive $400. If he is disabled a committee of three determines the extent of his injuries, and the length of time he shall receive a weekly stipend.

The company assumes charge of the funds and manages the accounts. The committees are from the miners.

At the Champion Mine the following method of relieving the injured is pursued. When a man is injured while in the employ of the company he receives $30 per month for his support. If he is killed outright or dies from his injuries, his widow or heirs receive from the company $500. To cover the expenses of this system the company charges each man 40c. per month and adds the same amount itself to the fund.

No man is required to pay assessments for medical attendance unless he desires to receive the benefits derived therefrom. A payment of 75c. per month for single men and $1.25 per month for married men entitles them to all the attendance and medicine that they may require for themselves and families.

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The Lake Superior Company assesses each man 30c. per month and contributes an amount equal to the aggregate of these assessments. Employés injured by accident are paid not less than $15 and not more than $25 per month, while under the care of the physician, for a period of not more than six months. Employés permanently injured receive a sum not exceeding $200. In case of death the administrator of the deceased receives $400 upon signing an agreement releasing the company from all further claims. The mine doctor receives $1 per month from every man.

Every man or boy about the Republic Mine is required to pay 15c. per month to the fatal accident fund, the company donating an amount equal to the aggregate of such assessments. Each year the company determines what amount can be paid to the representatives of each unfortunate during the ensuing year. For 1878 the sum was $400.

Each member of the Relief Club contributes 50c. per month, and if unable to work is entitled to $22.25 per month for not longer than twelve consecutive months. Members receiving relief must report to the mine doctor at least three times per week. According to the constitution of the Relief Club the mine captain is the president, the time-keeper is the secretary, and the Republic Iron Co. is the treasurer.

"All boys shall pay two-thirds the amount paid by the men, and will only receive two-thirds the benefit. Any member found intoxicated or habitually visiting saloons, or chopping or drawing wood or hunting for game, shall cease drawing benefits at once and his name be stricken from the books."

RULES AND REGULATIONS.

At many of the mines printed lists of rules and regulations are posted in conspicuous places, cautioning the men in regard to their conduct in the mine, and stating what is to be done in case of accident. In some cases these lists contain rules for the guidance of the men in drawing their pay. The following specimens have been selected as showing in a general way the character of the whole:

"All men, in the employ of this company or not, are forbidden to ride on any of the skips either ascending or descending, . . . as such skips are liable to get out of order and become dangerous.

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"All persons are forbidden to climb any ladder-way in any shaft while a skip is ascending or descending.

"Should any men be killed by a fall of ground, the place where such accident happens shall remain idle until it has been examined and secured by the officers of the mine, but all work in other places shall be resumed in the shift following that in which the accident occurred.

"In case of fatal accident men shall be allowed to quit work during the balance of the shift in which the accident occurred.

"Workmen under the influence of liquor will not be permitted about the work, and an offence of this kind will be considered sufficient ground for immediate dismissal, or fine, or both.

"All fines assessed against the men, except for damage to the property of the company, will be for the benefit of the Hospital fund."

ORE-SAMPLING.

By S. A. REED, Ph.D.

With the growth and multiplication of large custom works for the reduction of ores of the precious metals, has come sharp competition in the purchase of such ores. In the purchase of ores according to an established schedule, a prime necessity is an accurate estimation of the value of each lot—an estimation which shall satisfy both purchaser and seller. Where a mine reduces its own ores, very accurate estimation of the value of separate lots is seldom necessary—all that is required is a tolerably close notion of the working of the process, general percentage composition for the calculation of charges, and an approximate valuation of principal and by-products so as to minimize losses. But with a dozen concerns bidding for a single mine's product, as is often the case in Colorado, exact sampling must be assured, or a bid becomes merely a bet.

Again, a variety of causes have contributed to the strengthening of metallurgical works at large railroad centres, at the expense of works near the mines, so that it often pays better to ship ore of a certain character than to reduce it locally. The business of shipping ore has led to the establishment of a special class of mills called sampling-works, whose sole func-

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tion it is to determine correctly the coin value of lots of ore brought to them.

The sampler acts in one of several capacities. He is either a special agent or buyer for some particular metallurgical works at a distance—or he is an independent buyer, buying all lots coming to him, and shipping on his own account to the best market for the particular kind of ore in hand—or he is a broker to whom the miner ships his ore to be sampled and sold by the sampler, as the miner's agent, to the highest bidder. In any

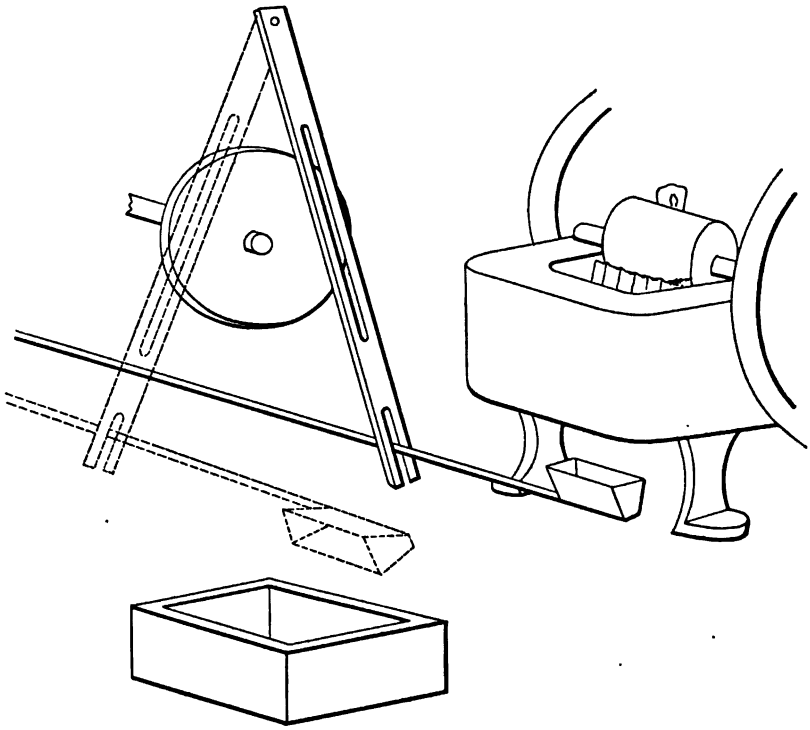

FIG. 1.

case the object is to make a determination which shall be first accurate ; second, simple and easily inspected ; third, cheap. Various methods of ore-sampling are in use, but it can be safely said that the subject has been more scientifically studied and carried nearer perfection in Colorado than anywhere else. By observing a few fundamental principles, almost any one of the sampling methods in vogue will give a good result, and a sampling-works does well occasionally to check its favorite method by some other.

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The particular methods vary in different works. Stamp-mills take samples from the batteries at stated intervals; some works take every tenth shovelful as a sample; others mix in a heap by raking the heap around with the shovel in a spiral, from the centre out, and then quartering, taking diagonally opposite quarters, remixing and requartering; others throw everything over a narrow scoop which catches a portion from each shovel; and finally, some adopt automatic devices for performing some one of these methods. The tendency is undoubtedly toward the general adoption of the scoop-shovel in some form. Automatic devices are excellent in some cases.

An automatic sampler used in a Leadville mill for taking intermittent samples directly from a Blake crusher is shown in Fig. 1.

A Georgetown mill has a sampling device, also on the intermittent principle, consisting of four cups, as in Fig. 2, caused to revolve slowly under a falling stream of ore, the cups discharging their samples into a receptacle at one side.

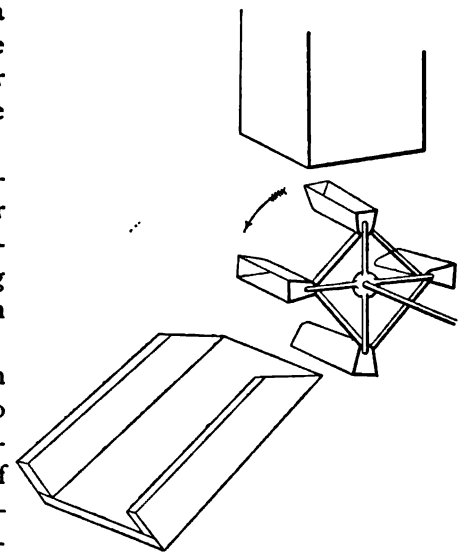

FIG. 2.

Prof. Egleston, in *Engineering* of December 15, 1876, describes several automatic sampling devices in use then in Colorado.

Some experience in sampling ores in Colorado has given me an opportunity of testing many of the methods by comparison, and I find the automatic arrangement I am about to describe, perfectly satisfactory. It is a counterpart of one previously used at a sampling-works in Clear Creek Co.

I pass every lot of ore first through a Blake crusher, set tolerably fine, and then through a pair of Cornish rolls, whence it is elevated by an endless belt with buckets to a revolving drum-screen with meshes $\frac{1}{8}$ inch square, what fails to pass the screen being returned to the rolls by a chute. The ore which passes the screen goes down the chute through the

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automatic sampler directly into a sack held at the mouth of the chute.

I have used a patent sack-fastener which can be tied up in a half a minute, and dispenses with sewing. One sack is tied while the next one is filling. The automatic sampler I use is shown in Fig. 3. (*a*) is a sheet-iron funnel by which the falling ore is thrown together into a small stream, whence it falls on (*b*) and is immediately scattered over the entire width of the box and falls into funnel (*c*), which, from the shape of its mouth, discharges a flat stream 12 ins. × 1 inch, and thoroughly mixed. The trough (*d*), 1 inch wide, is placed directly under (*c*) and at a sharp inclination, and carries off constantly one-tenth of the stream, the remainder going directly into the sack. The trough (*d*) leads by a tube into a box, and the portion conveyed there constitutes the sample.

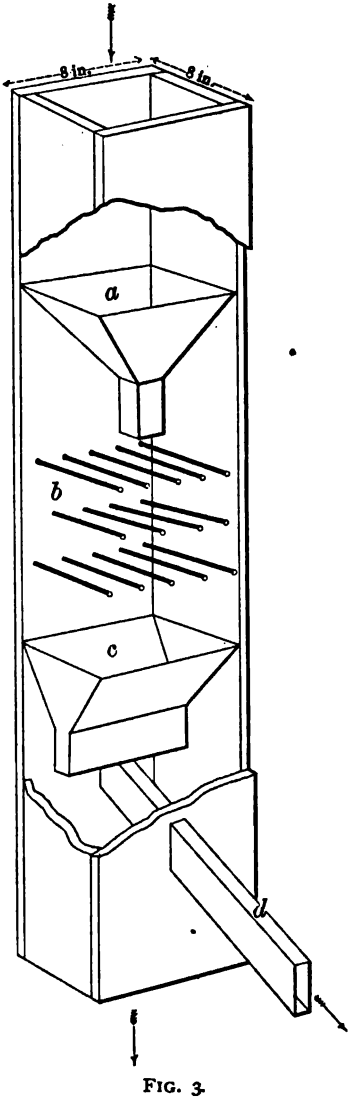

FIG. 3.

Thus when a 10-ton lot is run through the mill, a 1-ton sample is found in the box. This is then dumped on a clean floor and mixed by shovelling over a conical pile, every shovelful going to the apex of the pile. This is then thrown by shovelfuls over a scoop-shovel, constructed as in Fig. 4, or sometimes over one like Fig. 5. What is caught in the scoop, viz., about one-tenth of the main sample, is dumped alternately in two heaps, making a double sample, called original and duplicate. The two samples are then each cut down separately by throwing again over the scoop-shovel, sweeping up carefully each time, until they weigh about 10

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pounds each, when they are taken to a small pulverizer, constructed like a coffee-mill, and connected with the shafting. Here they are ground to about the size of coarse sand. Each

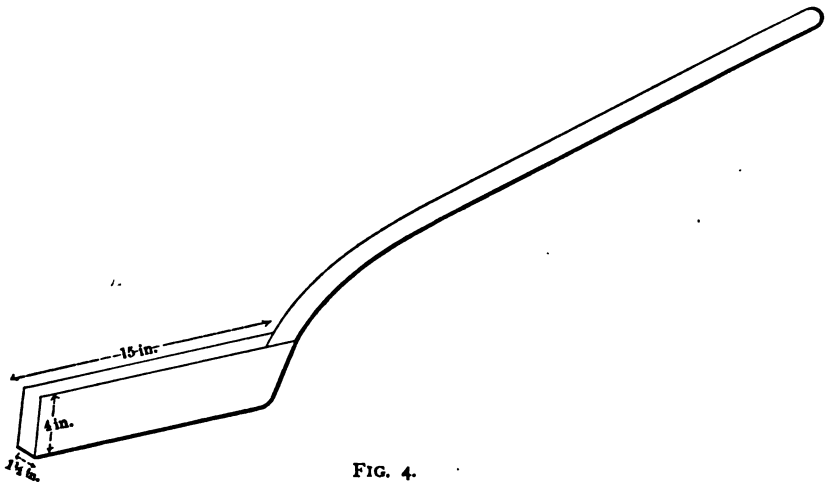

FIG. 4.

sample is then thoroughly mixed by rolling on an oil-cloth, and then cut down on an ordinary tin splitter, like Fig. 5—

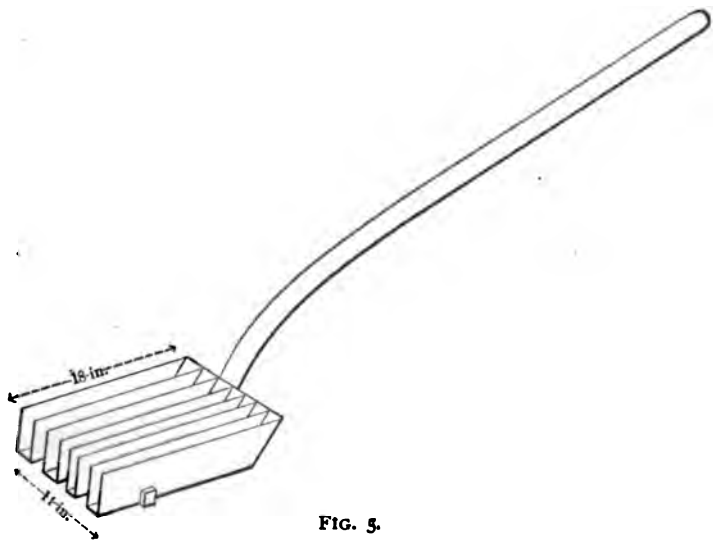

FIG. 5.

of rather deep and wide troughs, however—to about 2 pounds, when it is ground on a rubbing-plate till all passes a 20-mesh sieve. It is then cut down on a finer splitter with ½-inch

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troughs to about ¼ pound, and this is finally ground on the plate till all passes an 80-mesh sieve.

The two pulp samples are then mixed thoroughly on glazed paper, and each divided into three portions—one for the sampler, and one for the miner, who has it assayed by his own assayer as control, while the third is sealed up to serve as umpire in case of disagreement; buyer and seller have each an original and duplicate, and each is assayed in triplicate, so that twelve assays are made on the lot. The triplicates should agree well, to satisfy one of a correct assay; original and duplicate should agree, to satisfy one of a correct sampling; and the control must agree with the sampler, to satisfy the seller of fair dealing. A difference between sample and control of 1 per cent. on the whole value is usually settled by halving the difference, while a larger difference calls for a reassay and perhaps for a resampling of the entire lot.

In resampling, the whole lot is shovelled over the scoop-shovel on the floor of the mill. When both parties are satisfied, a settlement is made at schedule rates or the lot is put up for bids.

In some sample pulverizers the ground material falls directly over a set of segmental boxes grouped radially around the verterial shaft of the grinder. Seller and buyer then choose each a box for his sample.

Mechanical samplers are of limited use, however, in Colorado. Most of the smelters' sampling establishments sample by shovelling the entire lot over the scoop on the floor. At Salt Lake the quartering method is adopted. Where ores are to be smelted raw, as in Leadville, it is objectionable to have them brought to the fineness necessary for mechanical sampling. The practice then is to pass the lot through a Blake or Dodge crusher and through coarse rolls; to sample by scoop, and pass the large samples through finer rolls called sample-rolls; to reduce again with the scoop to a small lot which is put through a sample-grinder, and finally to a lot which is ground fine on a bucking-plate. Automatic samplers require the material to be crushed rather fine, and to be screened, lest coarse pieces clog the apparatus. For rich silver ores of irregular character, rather fine crushing gives a more accurate sample. Ore carrying native silver, coarse, is almost impossible to sample, but I have found that moderate quantities of wire silver disappear on the bucking-plate into

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fine powder which is quite evenly disseminated. Grinding in a mortar does not produce the same effect. Wet ore cannot be sampled, but ore need only be dried sufficiently to prevent adherence and packing.

It would be an interesting problem to work out mathematically from the doctrine of chances the principles of good sampling of an irregular mixture.

The points to be observed are:

First.—That there must be a certain maximum limit in the sample to the ratio between coarseness and quantity; for example, a panful of ore containing pieces as large as an orange would scarcely be a safe sample of a lot of ore.

Second.—There must be a minimum limit to the ratio between size of a direct sample and the entire quantity from which it is taken.

Third.—The method of taking the sample must be impartial in its action and free from any tendency to select one set of particles at the expense of others; for example, from ten tons of rich ore of the coarseness of gravel it would not be safe to take a first sample weighing only a few ounces. Further diminution must be preceded by further comminution.

Fourth.—Thorough mixing must be effected, and by a method which is free from any sorting action.

By pulverizing an entire 10-ton lot to 80-mesh fineness and thorough mixing, one might take a 10-ounce sample directly, but the aim is to minimize the crushing, hence one proceeds by successive steps. To take a $\frac{1}{10}$ sample from a 10-ton lot I might take the first step by simply crushing till no piece was larger than $2\frac{1}{2}$ inches diameter; but other conditions impel me to grind further, viz., the saving of wear on sacks where hard quartz is abundant, and the requirements of my automatic sampler.

All sampling processes have to be conducted with a view to guard against fraud, while granting full opportunity to the miner to convince himself that he is fairly treated.

The double sample is a great protection against tampering from outside, while, on the other hand, the miner is invited to witness every stage of the process while his lot is being sampled. One sampling-works in Colorado has a room with glass sides in the centre of the building, in which the miner sits and watches his ore sampled without being able to touch it.

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The practical working of the sampler I use is attested by the fact that with a season's average of over 150 ounces per ton, the lots shipped have been in every instance resampled at the smelter by scoop-shovel, and the differences rarely amount to 1 per cent.

DETERMINATION OF ZINC AS PYROPHOSPHATE.[1]

By GEORGE C. STONE, PH.B.

SOME time since I had occasion to analyze a sample of water said to contain the tailings from an amalgamating mill. The analysis of the suspended matter only added up about eighty-eight per cent. I tried boiling the filtrate from the magnesia with KOH: this threw down a fine, crystalline, white precipitate; on making a qualitative examination of this, I found it contained ZnO and P_2O_5. On weighing the precipitate, and calculating the amount of ZnO it contained, supposing it to be $Zn_2P_2O_7$, I found it eleven and a fraction per cent., which made my analysis add up very close to one hundred per cent. This led me to experiment with phosphate of zinc with the following results:

If a mixture of an acid solution of a zinc salt and an alkaline ortho-phosphate is exactly neutralized with ammonia, the zinc is precipitated as a bulky, flocculent, double phosphate of zinc and ammonia, easily soluble in acids or ammonia, and slightly soluble in potassic hydrate. If the solution in ammonia is boiled, the double phosphate reprecipitates as a fine, crystalline, white precipitate, very slightly soluble in ammonia, and almost absolutely insoluble in water. If the boiling is continued till the free ammonia is all driven off, the precipitation is complete. On drying and igniting this precipitate, the zinc is obtained as pyrophosphate, $Zn_2P_2O_7$.

This method of determining zinc has several advantages over the one commonly employed of precipitating as sulphide, dissolving in hydrochloric acid, reprecipitating as basic carbonate, igniting and weighing as oxide.

First.—It is much quicker, as it is not necessary to let the precipitate stand, and there is only one precipitation.

Second.—The precipitate settles rapidly, filters quickly, does not run through the filter, washes easily, dries quickly,

[1] A paper read before the American Chemical Society, March 3, 1882.

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brushes off the paper easily and completely when dry, and is very difficult to reduce.

Third.—The presence of ammonia salts does not interfere with the completeness of the precipitation. In all of which particulars its behavior is quite the reverse of that of either the sulphide or basic carbonate.

Fourth.—The pyrophosphate contains only 42.76 per cent. of zinc, while the oxide contains 80.25 per cent., so that an error in weighing makes much less difference in the result.

Fifth.—The phosphate does not need any purification, whereas the oxide frequently contains silica.

On the other hand, the phosphate bumps badly while boiling out the ammonia, and the precipitate sticks pretty tightly to the beaker. If the solution is kept rather dilute and only just enough ammonia is added to redissolve the precipitate first formed and the solution is frequently stirred, the bumping will not give much trouble. The precipitate that sticks to the beaker is best removed by dissolving in a few drops of HCl, adding a little $NaNH_4HPO_4$, making just alkaline with NH_4OH and boiling till the free ammonia is expelled.

Experiment I.—Was made more to test the behavior of the phosphate of zinc, and to see if the results agreed, than to test the absolute accuracy of the method. 0.2 gms. of zinc coated with oxide were dissolved in H_2SO_4 and a few drops of HNO_3, the solution was diluted to about 300 c.c., an excess of $NaNH_4HPO_4$ added, and just enough NH_4OH to redissolve the precipitate first formed. The solution was then boiled till there was no longer any smell of NH_3, filtered, washed thoroughly with hot water, dried, ignited and weighed. Gave:

A. $Zn_2P_2O_7$ 0.4627 gms. = Zn 0.1979 gms.
B. $Zn_2P_2O_7$ 0.4624 gms. = Zn 0.1977 gms.

A was filtered as soon as the NH_3 was expelled; B was allowed to stand in a warm place for three hours after boiling.

Experiment II.—0.2 gms. of zinc ore, mainly oxide, was dissolved in HCl, HNO_3, and H_2SO_4, evaporated till the HCl and HNO_3 were expelled, diluted, filtered, the iron separated as basic acetate, dissolved and reprecipitated by a large excess of NH_4OH, the filtrates combined and evaporated to about 300 c.c., an excess of $NaNH_4HPO_4$ added and the zinc determined as in *I.* In a second portion, of 1.0 gm., the iron was separated as above, the zinc precipitated by H_2S in a boil-

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ing acetic acid solution, let stand, filtered, washed, dissolved in HCl, reprecipitated as basic carbonate by Na_2CO_3, ignited and weighed as oxide. Gave :

A. $Zn_2P_2O_7$ 0.3441 gms. Zn 0.1471 gms. =73.55 per cent.
B. ZnO 0.9149 gms. Zn 0.7341 gms. =73.41 per cent.

Experiment III.—0.2 gms. ferruginous blende treated as in *II.*, A, gave :

A. $Zn_2P_2O_7$ 0.2721 gms. =Zn 58.17 per cent.
B. $Zn_2P_2O_7$ 0.2716 gms. =Zn 58.07 per cent.

The analysis was afterward completed and gave :

Zn (average of two determinations)	58.12 per cent.
Fe (" ")	7.61 per cent.
Insoluble silicious matter	1.43 per cent.
S calculated to give FeS and ZnS	32.96 per cent.
Total	100.12 per cent.

Experiment IV.—1.0 gm. $Zn(C_2H_3O_2)_2 2H_2O$ dissolved in water, excess of $NaNH_4HPO_4$ and NH_4OH added, the NH_3 expelled by boiling and the zinc determined as before, gave :

A. $Zn_2P_2O_7$ 0.6953 gms. =Zn. 29.73 per cent.
B. $Zn_2P_2O_7$ 0.6974 gms. =Zn. 29.82 per cent.

The theoretical percentage is 29.68.

Experiment V.—To test the influence of alkaline and ammonia salts. 0.5 gms. $Zn(C_2H_3O_2)_2 2H_2O$, 2.0 gms. Na_2CO_3, 2.0 gms. NH_4Cl, and 5.0 gms. $NaC_2H_3O_2$, the zinc determined as before, gave :

A. $Zn_2P_2O_7$ 0.3484 gms. =Zn 29.78 per cent.
B. $Zn_2P_2O_7$ 0.3469 gms. =Zn 29.66 per cent.

Experiment VI.—As the phosphate of zinc is soluble in NH_4OH, while the phosphate of lime, magnesia and manganese are insoluble in it, it seemed probable that the zinc could be separated by means of this reaction. The following experiments were tried to test this : 0.5 gms. $Zn(C_2H_3O_2)_2 2H_2O$, 0.1 gm. Mn_3O_4, 0.1 gm. $CaCO_3$ and 0.1 gm. $MgSO_4 7H_2O$ were dissolved in HCl, excess of $NaNH_4HPO_4$ and NH_4OH added and let stand cold for eighteen hours, the clear solution filtered off, the precipitate dissolved in HCl, reprecipitated by NH_4OH and a few drops of $NaNH_4HPO_4$, filtered, washed with cold dilute NH_4OH, the NH_3 boiled out and the zinc determined as before, gave :

A. $Zn_2P_2O_7$ 0.3444 gms. =Zn 29.46 per cent.
B. $Zn_2P_2O_7$ 0.3451 gms. =Zn 29.52 per cent.

The phosphates of manganese and lime clogged the filter

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and showed a tendency to run through it. To get good results the solution must be rather dilute (400 to 500 c.c.).

Experiment VII.—0.5 gms. $Zn(C_2H_3O_2)_2 2H_2O$ and 0.5 gms. $MgSO_4,7H_2O$, treated as in the last experiment, gave:

A. $Zn_2P_2O_7$ 0.3453=Zn 29.54 per cent.
B. $Zn_2P_2O_7$ 0.3451=Zn 29.52 per cent.

Experiment VIII.—As phosphate of lime filters very slowly and is very difficult to wash, I tried precipitating the lime as oxalate in a rather dilute solution, decanting, dissolving in HCl, reprecipitating and determining the zinc in the filtrate as phosphate. 0.5 gms. $Zn(C_2H_3O_2)_2 2H_2O$, and 0.25 gms. $CaCO_3$ taken, gave:

$Zn_2P_2O_7$ 0.3429 gms. =Zn 29.32 per cent.

Experiment IX.—1.365 gms. Zn_2P_2O, heated to redness for about half an hour weighed 1.365 gms.

The results of the separation are not as good as I could wish; but I intend to continue the experiments in hopes of improving on them.

RÉSUMÉ.

Experiment.	Substance analyzed.	Grammes used.	Grammes zinc found.	Remarks.
I.	Impure zinc.	0.2	0.1979	
	" "	0.2	0.1977	
II.....	Zinc ore.	0.2	0.1471	Weighed as $Zn_2P_2O_7$.
	" "	1.0	0.7341	Weighed as ZnO.
III....	Blende.	0.2	0.11634	
	"	0.2	0.11614	
IV....	$Zn(C_2H_3O_2)_2 2H_2O$	1.0	0.2973	Theory requires 0.2968 gms. Zn.
	"	1.0	0.2982	
V.....	"	0.5	0.1489	+2 gms. NH_4Cl+2 gms. Na_2CO_3+5 gms. $NaC_2H_3O_2$.
VI....	"	0.5	0.1484	
	"	0.5	0.1473	+0.1 gm. Mn_3O_4+0.1 gm. $CaCO_3$+0.1 gm. $MgSO_4,7H_2O$.
	"	0.5	0.1476	
VII...	"	0.5	0.1477	+0.5 gms. $MgSO_4,7H_2O$.
	"	0.5	0.1476	
VIII..	"	0.5	0.1466	+0.25 gms. $CaCO_3$ separated as oxalate.
IX....	$Zn_2P_2O_7$	1.365	1.365	Ignited for half an hour.

Since making the above experiments I have found that Hugo Tamm (*Chemical News*, xxiv., p. 148) has proposed to weigh zinc as the double phosphate of zinc and ammonia, $Zn NH_4PO_4$, which, of course, necessitates the use of a weighed filter. His method is as follows: Make acid solution of a zinc salt alkaline with NH_4OH, then barely acid with HCl and add a solution of Na_2HPO_4, heat nearly to boiling for a few minutes, filter, wash, dry at 100° C., and weigh as $ZnNH_4PO_4$.

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He states that there is invariably a loss when the double phosphate is converted into pyrophosphate by ignition; but from the results of my own experiments I should say the loss was due to his precipitate being formed in an acid solution, in which I have found the double phosphate to be quite soluble. The greater quantity of zinc found when weighing the double phosphate was probably due to insufficient heat in the drying. He uses the atomic weight of 65.5 for zinc; using 65, the one now generally accepted, his results would all be low.

The composition of pyrophosphate of zinc is as follows:

		Per cent.
2Zn	130	42.763
2P	62	20.395
7O	112	36.842
$Zn_2P_2O_7$	304	100.000

or

2ZnO	162	53.29
P_2O_5	142	46.71
$Zn_2P_2O_7$	304	100.00

Nickel and cobalt form phosphates which behave precisely like the corresponding zinc salt; both, however, are strongly colored, and so can readily be distinguished from phosphate of zinc.

SAN JUAN MINE.

By F. D. BROWNING.

THIS mine is located in the district of Cieneguilla, probably the oldest mining district in the State of Chihuahua. It is about sixty miles west of the City of Chihuahua, to which, as well as to all the neighboring places, there are good wagon roads. Its elevation is about 7,000 feet, and the climate superb. The whole district and town of Cieneguilla are traversed by a stream of good water throughout the entire year, and by numerous streams during the rainy season. The supply of wood for fuel, and pine timber for lumber and use in the mines, is abundant and at hand.

The mines of this district are on true fissure-veins, traversing the southern slope of a high range of mountains, and having the same general trend as the latter. There seem to be two

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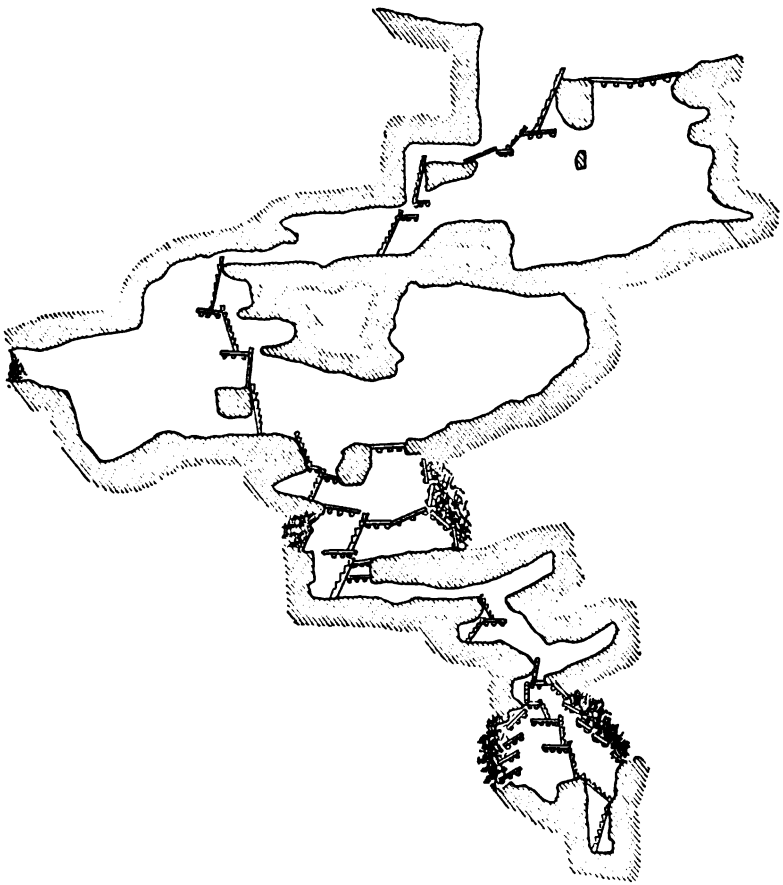
distinct mineral belts on the slope; the one higher up containing veins which are rich in silver, while in the lower one are found veins charged with lead ores carrying little or no silver.

Going south from the mines, about two miles down the stream, to the point where the latter emerges from the foothills, we come to the town. This comprises a few houses of



SAN JUAN MINE.

adobes, or stone and mud, and the ruins of a once large village, located at the head of a beautiful valley.

History.—In 1691 the mines and town were abandoned on account of the Apache Indians, and remained so till 1716, when the San Juan and La Deseubridora mines were denounced by a Spanish grandee. From these mines he took a large fortune, acquired the haciendas Bustillos, La Quemada, La La-

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guna, San Diego, and Vilches, including an area thirty leagues long and ten leagues wide, lying to the north of Cieneguilla, and built up a large town, having five churches, called Sta Vera Cruz de la Cieneguilla, which at that time was the capital of the Department of Chihuahua, and the most important town in the Territory. Ores which he took from these mines were beneficiated in Barraganes, La Concepcion, San Juan de Chivirsé, San Gregorio, and El Mastranzo, small towns situated from three to ten miles from Cieneguilla, as well as in Cieneguilla itself. Reduction was effected by smelting, in the patio, and in copper kettles. Large amounts of the slag were carried to Cusihuiriachic, twenty miles distant, and sold for $13 per ton. He worked till a date not definitely known, when the Apaches again depopulated the town.

In 1796 the same mines were again denounced by one Armendáriz, who worked them with great success for many years. But in 1832 the town, as well as all the surrounding country, was again devastated and depopulated by the Apaches, those hounds of hell who have ever fought civilization and industry to the bloody end—the wholesale butchers of mankind, who are, nevertheless, most tenderly nursed and cared for by the United States Government.

In 1856 the same mines were denounced by one José Gregorio Anchonda, who worked them till May, 1881, when some Boston gentlemen purchased them, together with a number of other mines in the same district.

The vein in which the San Juan Mine is worked cuts across one of the foot-hills of the above-mentioned mountain range, in a course N. 85° W., dipping 75° to the S. The vein is traceable for half a mile, in which distance are several old mines more or less worked, but by far the most extensive and important of these is the San Juan.

The vein has an average width of 4 feet at the surface, and for the most part is much decomposed. The silver is mostly in the form of green chloride, and the ore is very docile.

The lowest works now accessible in the San Juan Mine are about 275 feet deep, while the lateral extent is about 230 feet, but all as irregular as were the deposits. In many places the passage-ways are entirely filled, hence the extent of the old workings cannot be ascertained. The mine is perfectly dry, and even 200 or 300 feet deeper, where the vein was proved by another mine, no water was found. Throughout this mine

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the vein has been strong, and not a few places bear unmistakable evidence of bonanzas. Here and there a rich piece of ore can be found remaining on the walls, but very few of these have escaped the practised eye of the Mexican "gambucinos" (mine robbers), and they alone can find these by candle-light. In places the walls are impregnated with chloride of silver to a depth of several feet. As a whole, this impregnated rock is very low grade, but by careful sorting and cleaning by hand, at which the Mexicans are very expert, a large amount of workable ore can be obtained from it.

The dump from this mine is large, and notwithstanding scores of men have picked it over for more than a century, and hundreds of people are said to have lived from it for several years, pieces can still be found in it whose cleavage-surfaces are spotted with the green chloride of silver.

The present company has done no new work in the mine except to sink a winze in the bottom, which is now 35 feet deep.

Soon after this was commenced, a few hundred pounds of very rich ore were struck, pieces of which assayed from 954 to 1,700 ozs. per ton, but a "horse" followed immediately below, occupying nearly the full width (5 feet) of the vein.

Between this "horse" and both walls, which are well defined, occur dirt-seams from two inches to a foot thick, containing iron, sands carrying silver, and some chloride of silver, the whole assaying 17 ozs. per ton. I have learned that since my visit to the mine, a body of very rich ore has been struck in the bottom of this winze.

The whole vein is so soft at that point that it can be mined with a bar without blasting.

The ore, as well as waste rock, is carried to the surface on the backs of men; there it is sorted and cleaned, and thence taken to the reduction works in the town in a wagon drawn by mules.

Reduction of the Ore.—The ore which is not already sufficiently fine is broken to nut-size by hand, then ground in arrastres till the coarsest would pass a twenty-mesh sieve, then lixiviated by the Von Patera process, and the tailings from this concentrated on an inclined bed by hand, and the concentrations sold to a smelting company.

This is rather a singular combination of primitive and modern scientific processes, but the Mexican processes which

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at first seem so crude should not be too quickly ignored and condemned; for while they cannot compete with improved machinery where labor is high, fixings and repairs are cheaply obtainable, and large quantities of ore have to be promptly reduced, they can be used successfully and profitably where machinery could not be thought of—for example, in out-of-the-way places where the mines are not developed, and the quantity and quality of ore not certainly ascertained. Until these two points are settled beyond a doubt, no large amount of money should be expended for reduction works.

The cost for breaking the ore by hand between two stones is $1.05 per ton, and when we consider that not a cent is to be added to this for materials or repairs, it is not an extravagant method for crushing moderate quantities of ore. While arrastres are not as well adapted for grinding ores to be lixiviated as those to be amalgamated, because a quantity of very fine slime is produced which retards leaching, nevertheless they can be and are successfully used for this purpose; and the cost for grinding in them is only $1.00 per ton.

The Von Patera process is the one naturally adapted to the extraction of the chloride of silver; because no expensive plant is required, tanks of masonry lined with pine-tar being used; all the materials for making the solutions can be easily obtained; and especially because it is cheap, only costing $1.50 per ton.

The method of concentrating the tailings is primitive, but perfect. The concentrator consists of a bed of masonry, about 6 feet long and 4 feet wide, inclined about 10°, with sides about 8 inches high. The lower end of this reaches to the edge of a running stream of water.

The tailings are shovelled in a pile on to the upper part of the bed. A Mexican sits on a plank thrown across the stream opposite the lower end of the bed, and with a horn, with a slit cut from the side to allow the air to escape, directs a stream of water, which is kept nearly constant, first against the foot of the pile, which washes the ore down the incline, then against the sheet of ore and water running down, which separates very perfectly the heavy particles from the light, the former settling to the bottom and being held by a cleat across the lower end, while the latter are washed over into the stream.

This concentrator is called a "planilla."

The cost for concentrating one ton to 200 lbs. is $1.60, and

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the result of the work can best be shown by the following record of assays:

Ore before leaching, 14 ozs. per ton.

One ton of same concentrated to 250 lbs., 110 ozs. per ton.

[NOTE.—The small loss was occasioned by scales of chloride of silver being carried off by the water; because the tailings from concentration assayed nothing.]

Ore after leaching with CaS_2O_3 in the tank, 6 ozs. per ton.

One ton of same tailings concentrated to 166 lbs., 65 ozs. per ton.

Tailings from concentration, 0 ozs. per ton.

The following statement of costs will serve to show how economically work may be performed, even when working in a very small way and with Mexican methods:

Cost to mine and extract 1 ton of ore	$3.50
" haul to reduction works 1 ton of ore..	.40
" break 1 ton of ore.....	1.05
" grind 1 ton of ore.....	1.00
" lixiviate 1 ton of ore.....	1.50
" concentrate 1 ton of ore.....	1.60
Total.....	$9.05

The accompanying sketch of the San Juan Mine is a fair representation of a Mexican mine.

The workings are about as irregular as in most mines, and the facilities for entering, leaving, and extracting ore—by pole-ladders—are the usual ones.

MINE SURVEYING.

BY G. A. SUTER AND H. S. MUNROE.

OWING to the difficulties which are generally experienced in the connection of underground with surface surveys, and in carrying the lines of underground surveys through winzes and stopes of steep pitch, we hope that the following review of the methods with which we are acquainted may prove interesting to our readers. When the territory of the underground survey, for example a mine, is connected with the surface by an horizontal heading or by an inclined shaft, the connection is easily made with an ordinary transit. When, however, access to the mine can be had only through a vertical shaft, perhaps

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not more than six feet square and several hundred feet deep, the problem becomes more difficult. Four different methods may then be employed, each one having its advantages under peculiar circumstances. The first and second, although pretty generally known, will still bear repetition, while the third, which we take from a recent number of *Engineering*, and the fourth, suggested by Mr. S. W. Balch, '83, S. of M., we believe to be new.

First Method: With a Mining Transit.—In a transit of ordinary construction the telescope cannot be elevated or depressed more than 50° without the line of sight being intercepted by the horizontal limb of the instrument. In a mining transit this difficulty is overcome by several different forms of construction. In the eccentric mining transits an auxiliary telescope is attached outside of the standards to the prolongation of the horizontal axis of the main telescope, or the main telescope itself is permanently mounted in a similar position.

Another plan is to incline the standards of the telescope forward, so that its axis overhangs the horizontal limb, and when the telescope is brought into a vertical position the line of sight passes in front of the plates. The same result is obtained by mounting an auxiliary telescope on standards on top of the main telescope, so that it moves with the main telescope and in the same vertical plane. By the use of any of these instruments, especially if the telescopes be provided with a diagonal eye-piece, the transfer of a line down a shaft presents little greater difficulty than the sighting with an ordinary instrument down a steep hill. Great care should, however, be taken that the instrument is in perfect adjustment, so that the telescope shall revolve in a vertical plane. Sometimes, to avoid error from this source, two points in the survey-line are located at the bottom of the shaft, and the instrument being set over one of these, a back sight to the other serves to check the back sight taken on the station at the surface. The second point at the bottom may also be determined with the aid of a single plumb-line, as follows: The transit being set up on one edge of the shaft, at the top, a straight-edge is carefully placed in the line of sight, near the end of which a plumb-line is attached, which extends to the bottom of the shaft. A point at the bottom of the shaft, on the side away from the transit, has been previously sighted at and marked. Having done this, the transit is taken to the bottom of the shaft and placed in

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line with the station and the wire. The line thus determined will correspond to the line on the surface, and may be used as a base for the underground survey. In suspending the plumb-line, the precautions mentioned in the next method should be observed.

Second Method: By Two Plumb-Lines.[1]—In this a line is run across the top of the shaft, projected to the bottom by means of two plumb-bobs, and there used as a new base line. The line across the shaft is marked in the timbers by tacks or by cleats, which may be permanent and used in future surveys if the earth about the shaft is sufficiently firm. A straight-edge is placed against the tacks and two plumb-bobs, weighing about ten pounds each, suspended from it by means of annealed iron wire. The bobs are received in buckets of water or mud, in order to stop or lessen their vibrations, which result from four causes: (1) the pendulum-like motion received while being lowered; (2) the drippings of water from the sides of the shaft falling upon the wires; (3) the influence of air-currents; and (4) the difference in the earth's velocity at the bottom of shaft and at the surface is also said to produce vibration in so long a pendulum. The vibration or displacement of the wires resulting from air-currents can be more effectually stopped by enclosing them in boxes at all points where they are so affected.

When the plumb-bobs have become stationary or nearly so, a transit is set up in line with the wires. This is accomplished by changing the position of the instrument until the nearer wire coincides with the vertical hair and the second wire is concealed by the first. Or, if the wires will not come entirely to rest, the transit will be in line when the amount of vibration of second wire to one side of the first is equal to the extent of the vibration on the other side. The greater the distance between the wires, and the farther the instrument is from the wires, the more accurately can it be placed in line. The straight-edge at the mouth of the shaft should therefore be placed so that the greatest possible distance between the wires and the longest range at the bottom shall be obtained. When the transit is in line, two points on the line should be projected up into the timbers by plumb-lines, and marked by spuds, which are made by punching a hole into the flat head of a horseshoe-nail. By this means the plumb-bobs are always

[1] See also, Shaft Surveying, by Ellis Clark, Trans. Amer. Inst. of Mining Engineers, vol. vii., p. 139.

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hung in exactly the same position. Care should be taken that the wires do not come in contact with the sides of the shaft or with a projecting timber. To insure this a lamp is slowly passed around the wire at the bottom. If this light can be seen at the top of the wire in whatever position it is placed, then the wire is clear. It is best also to work at night, when the unsteadiness of the plumb-bobs will not be increased by the jar of hoisting machinery or the tramping of miners.

In surveys made by E. Clark, of Easton, Pa., the base lines varied from 6″ to 18″, and even with these small bases he obtained very good results when the underground survey was not too extended.

Third: The Severn Tunnel Method.—This novel method was employed by Chief Engineer Richardson in directing the heading driven as the forerunner of the railway tunnel under the river Severn. On account of the length of the heading, the incessant jar of the pumping-engines, and the extreme wetness of the shaft, neither of the plumb-bob methods was applicable. The length of the proposed heading was two miles, and the available base line 12 feet. An error in observation of $\frac{1}{30}$ of an inch, therefore, would have become 45 inches at the end of two miles. As the probable error would exceed $\frac{1}{20}$ of an inch, it became necessary to adopt some method which would not involve the use of plumb-bobs. A large transit, in which the telescope could revolve completely round on its horizontal axis, was firmly set up over the shaft and accurately in the vertical plane passing through the centre line of the tunnel. This line was determined by two sight-poles, one on each side of the river. The heading having been driven a short distance, a horizontal wire, 100 yards long, was stretched at the bottom of the shaft. One end, A, was attached at the side of the shaft farthest from the heading, the other, B, at a point 100 yards along the heading. About 14 feet of wire, which was illuminated by an electric light, was visible from the top of the shaft. The ends of the wire, A and B, were passed over the V-threads of horizontal screws and stretched by means of weights suspended from the ends. Thus, by turning either screw, a very slight lateral motion could be imparted to the corresponding end. The telescope having been carefully set up and levelled as stated above, the end A was sighted at and the corresponding screw turned until this end was brought exactly in the centre line. The telescope was

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then directed toward the farthest point of the wire visible on the other side of the shaft, and this point also brought into the centre line by turning the screw at the end B, 100 yards distant. Thus not only the 14 feet visible but the entire length of the wire became accurately directed into the line of the tunnel. If there was no refraction, due to differences of temperature of the air in the shaft, the only error which could occur would be the very minute error of observation of the position of the wire. Even if the transit had been slightly out of level the result would not have been materially affected. For it is obvious that, if the instrument had been tilted about an axis perpendicular to the direction of the tunnel, then the plane described by the vertical cross-hair would still have been vertical. Again, if the telescope had been slightly tilted about an axis parallel to the tunnel, the vertical cross-hair would have described an oblique plane. But if the wire at the bottom of the shaft and the line sighted on the surface were both horizontal, then, when the former was brought in line, they would represent the intersections of the oblique plane described by the telescope with two horizontal planes, and would, therefore, still be parallel to each other. The line at the bottom would therefore have been shifted parallel to its proper position, an error which would not have been multiplied at the far end of the heading. A base line 100 yards in length and practically free from error was thus obtained. The headings are said to have met *exactly*, not with a difference of three inches, as was at first reported. No better test of the accuracy of this method could be desired.

Fourth Method: With a Tilted Transit.—This method may be used with any transit provided with a vertical arc or circle and along level under the telescope. The transit is inclined sufficiently to permit a full view of the bottom of the shaft through the telescope. The instrument is so adjusted, with the aid of the levelling-screws, that the centres shall be inclined in the direction of the line to be prolonged down the shaft, which ensures that the telescope shall revolve in a vertical plane. This adjustment can be made approximately by means of one of the short bubble-tubes on the plates—the one which is at right angles to the line of sight being levelled, the telescope at the same time being kept in the proposed line of sight. The final adjustment would be most conveniently made with the aid of a striding level placed on the

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axis of the telescope. As but few instruments are provided with such an attachment, some other means must be employed. The long level under the telescope may be used for the purpose as follows: Set the horizontal limb at 0°, the telescope being in the line of sight, and level the telescope accurately; then by means of the tangent-screw turn the instrument 5° and observe the deviation of the bubble; next swing the instrument an equal amount on the other side of the line of sight and correct half the error with the levelling-screws, using the pair of screws at right angles to the line of sight. Set the plates again at zero and repeat till the bubble shows the same deviation on each side. The reading on the vertical circle will then be equal to the inclination of the instrument, and the telescope will then revolve in a vertical plane and may be used to transfer the surface line to the bottom of the shaft. The back sight from the bottom of the shaft may, of course, be taken with the instrument in its normal position, if the telescope is provided with a diagonal eye-piece.

It is often necessary, however, to turn an angle in going down the shaft, as, for example, when surveying from level to level through winzes or stopes. In such case, the instrument is set up as before, with the centres inclined toward the lower station, and the vernier of the horizontal limb set at zero. Note the inclination of the instrument, and then sight to the upper station, reading both horizontal limb and vertical circle. The angle as read on the instrument in its inclined position may be reduced to a horizontal angle by the following formula:[1]

$$H = tan.^{-1}\left\{\frac{tan.\ \frac{1}{2}\ h \times cos.\ \frac{1}{2}\ (90° + v - i)}{cos.\ \frac{1}{2}\ (90° + v + i)}\right\} + tan.^{-1}\left\{\frac{tan.\ \frac{1}{2}\ h \times sin.\ \frac{1}{2}\ (90° + v - i)}{sin.\ \frac{1}{2}\ (90° + v + i)}\right\} \qquad (1.)$$

in which H = true horizontal angle; h = angle read on vernier plates; v = angle read on the vertical circle; and i = the inclination of the centres from the vertical, or of the plates with the horizon, determined as above.

In case the instrument can be placed so that $h = 90°$ and $v = 0°$, then H will also be 90°, and no correction will be required.

[1] *Tan.*$^{-1}$ = "the arc whose tangent is" { }.

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In case it should be desirable, or more convenient, not to incline the axis of the instrument in a definite direction, as above, it is then only necessary to set the vernier at zero when the telescope is in the meridian in which the instrument is inclined, sight at each station, and correct each reading as referred to the above meridian by the formula. To find the angle between the two stations, the angles between each and the meridian plane found as above are added to or subtracted from each other according as the stations lie on opposite sides or on the same side of the meridian plane.

The preliminary adjustment of the instrument is made very much as before. The vernier of the horizontal limb being set at zero, the lower clamp is loosened, and the instrument turned on its outer centre till the small bubble-tube shows that the instrument is approximately in position. The telescope is now turned 5° in each direction by means of the upper tangent-screw, and the deviation of the bubble in the telescope-level noted in each case. If this deviation has been to the same amount, and in the same direction, then the plates are in proper position. If not, then one-half the error is corrected by the lower tangent-screw, and the operation repeated until the instrument is in adjustment.

A practical test of the method was made by Mr. Balch, the same angle was measured carefully with the instrument in its normal position, and again with the tilted instrument, by the method last described. The difference by the two methods was about two minutes, a result not altogether satisfactory, but which may perhaps be bettered upon further trials. Mr. Balch calls attention to the fact that, in the inclined position, the *centre of the telescope axis* should be placed over the station, instead of the point where the plumb is usually attached.

The vertical angle V to any station may be obtained by the following formula:

$$Cos.\ V = \frac{sin.\ h.\ cos.\ v.}{sin.\ H} \qquad (2)$$

DEMONSTRATION OF FORMULÆ.—Let M A N represent a sphere whose radius is unity, circumscribed about centre of the telescope; A O L, the intersection of the meridian plane with the surface of the sphere; M O G, the intersection of a

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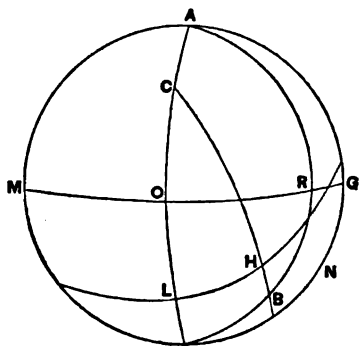
horizontal plane passing through the centre; L H, the arc of a great circle parallel to the limb; B, the intersection of the line of sight; A, the pole of M G; C, the pole of L H; A B and C B, arcs of great circles.



Then will O L=the angle of inclination of the limb=i; L H =the angle read on the limb= h; H B=the angle read on the vertical circle=v; O R=the true horizontal angle=H; R B =the true vertical angle=v.

In the triangle A B C, A= H; C=180°−HCL=180°−h; AC=OL=i; CB=CH+HB =90°+v.

∴ cot.$\frac{1}{2}$C=tan.$\frac{1}{2}h$; $\frac{1}{2}$(CB+AC)=$\frac{1}{2}$(90°+v+i); $\frac{1}{2}$(CB−AC)=$\frac{1}{2}$(90°+v−i).

∴ From the first set of Napier's Analogies we have

cos. $\frac{1}{2}$(90°+v+i) : cos. $\frac{1}{2}$(90°+v−i) :: tan. $\frac{1}{2}h$: tan. $\frac{1}{2}$(H+B)

sin. $\frac{1}{2}$(90°+v+i) : sin. $\frac{1}{2}$(90°+v−i) :: tan. $\frac{1}{2}h$: tan. $\frac{1}{2}$(H−B)

$$\left.\begin{aligned} \therefore \text{tan. } \tfrac{1}{2}(\text{H}+\text{B}) &= \frac{\text{tan. } \frac{1}{2}h \times \text{cos. } \frac{1}{2}(90^\circ+v-i)}{\text{cos. } \frac{1}{2}(90^\circ+v+i)} \\ \text{tan. } \tfrac{1}{2}(\text{H}-\text{B}) &= \frac{\text{tan. } \frac{1}{2}h \times \text{sin. } \frac{1}{2}(90^\circ+v-i)}{\text{sin. } \frac{1}{2}(90^\circ+v+i)} \end{aligned}\right\} (3)$$

$$\tfrac{1}{2}(\text{H}+\text{B})+\tfrac{1}{2}(\text{H}-\text{B})=\text{H}.$$

$$\therefore \text{H}=\text{tan.}^{-1}\{\text{tan. } \tfrac{1}{2}(\text{H}+\text{B})\}+\text{tan.}^{-1}\{\text{tan. } \tfrac{1}{2}(\text{H}-\text{B})\}$$

substituting the values of tan. $\frac{1}{2}$(H+B) and tan. $\frac{1}{2}$(H−B), obtained from (3), we get formula (1), as already given.

In the same triangle,

$$\text{sin. A} : \text{sin. C} :: \text{sin. CB} : \text{sin. AB}.$$

But sin. A=sin. H; sin. C=sin. (180°−h)=sin. h; sin. CB=sin. (90°+v)=cos. v; sin. AB=sin. (90+ V)=cos. V.

$$\therefore \text{sin. H} : \text{sin. } h :: \text{cos. } v : \text{cos. } V.$$

from which obtain formula (2).

In applying formulas (1) and (2), it will be found necessary to consider angles of depression on the vertical circle as positive; angles of elevation as negative.

Example.—Given i=31° 10′; v=10° 15′; h=52° 44′. In

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formula (1) calling the first term of the second member m, the second term n, and applying logarithms, we have :

log. tan. $\frac{1}{2} h$ (26° 22′)	9.695201
+ log. cos. $\frac{1}{2}$ (90° + $v - i$) (34° 32′ 30″)	9.915777
+ a . c . log. cos. $\frac{1}{2}$ (90° + $v + i$) (65° 42′ 30″)	0.385755
— 10.	10.
log. tan. m =	9.996733

$\therefore m = 44° 47′ 4″$

log. tan. $\frac{1}{2} h$ (26° 22′)	9.695201
+ log. sin. $\frac{1}{2}$ (90° + $v - i$) (34° 32′ 30″)	9.753586
+ a . c . log. sin. $\frac{1}{2}$ (90° + $v + i$) (65° 42′ 30″)	0.040260
— 10.	10.
log. tan n =	9.489047

$\therefore n = 17° 8′ 14″$

$H = m + n = 61° 55′ 18″$

Applying logarithms to formula (4), we get :

log. sin. h (52° 44′)	9.900818
+ log. cos. v (10° 15′)	9.993013
+ a . c . log. sin. H (61° 55′ 18″)	0.054382
— 10.	
log. cos. v =	9.948213

$\therefore V = 27° 25′ 41″$

SINKING OF SHAFT "B," BARNUM MINE, ISHPEMING, MICH.*

By R. H. VONDY, '82.

In the month of June, 1879, a diamond-drill hole was bored 750 feet west of the present location of shaft A, and 75 feet north of the location of shaft B. A thick bed of iron ore was reached at a depth of 266 feet, which, it was thought, could be worked profitably.

* Being an extract from the memoir of Messrs. Vondy and Emrich. This portion printed gives an interesting account of shaft-sinking under difficulties rather than a method of sinking to be followed under similar circumstances. Had these difficulties been anticipated from the start no doubt some other plan of work would have been adopted.—Ed.

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Having succeeded in sinking shaft A to a good depth, it was decided to sink another, for two reasons: first, that it would be more economical to work with two shafts than with one; and, second, that in case of any accident in the mine, or in one of the shafts, the miners would have the other to escape by.

The quartzite and the ore-deposits are here covered by a bed of sand, 102 feet thick. The first bed, which is of lean ore, is 170 feet 8 inches below the surface of the quartzite and is 3 feet 6 inches thick. The next bed, which is good ore, is at a depth of 242 feet 4 inches, and is only 8 inches thick, under which lies a bed of ore mixed with chlorite, which is 26 feet 10 inches thick, and under this again (261 feet 1 inch below the ledge) lies the bed of good ore, which is 26 feet 5 inches thick.

On January 26th, 1880, the company commenced the sinking. The ground was levelled off, and cedar logs, 40 feet long and 9 inches in diameter, were laid in two horizontal layers (the logs in one layer lying across those in the other) in such a manner as to leave a space in the centre about 14 feet square, then four large pine timbers, 16 inches square and 40 feet long were laid across these, the second two lying at right angles with the first two, leaving a clear space in the centre 12 feet square.

The forty feet square of cedar logs was intended as a foundation for the shaft-house and also to support the large cross-timbers from which the timbering of the shaft was suspended.

The shaft-house is 19 feet 6 inches square at the base, 12 feet square at the top, and 22 feet high. It consists mostly of 12-inch by 12-inch timbers.

The sinking was commenced by driving 3-inch planks down into the sand, outside of the 16-inch timbers, and shovelling the sand from the inside space as the boards went down until a depth of about 5 feet was reached, when the first set of timbers was put in place. These were 14 inches square and 14 feet 4 inches long, and when put together left a space 12 feet square inside. The timbers were held in position by sixteen iron bolts, 1¼ inch in diameter and 6 feet 6 inches long, four to each timber. They went through holes in the large foundation timbers and through holes in the first set below, when the washers and nuts were put on and the timbers drawn up until they were squeezed firmly against four

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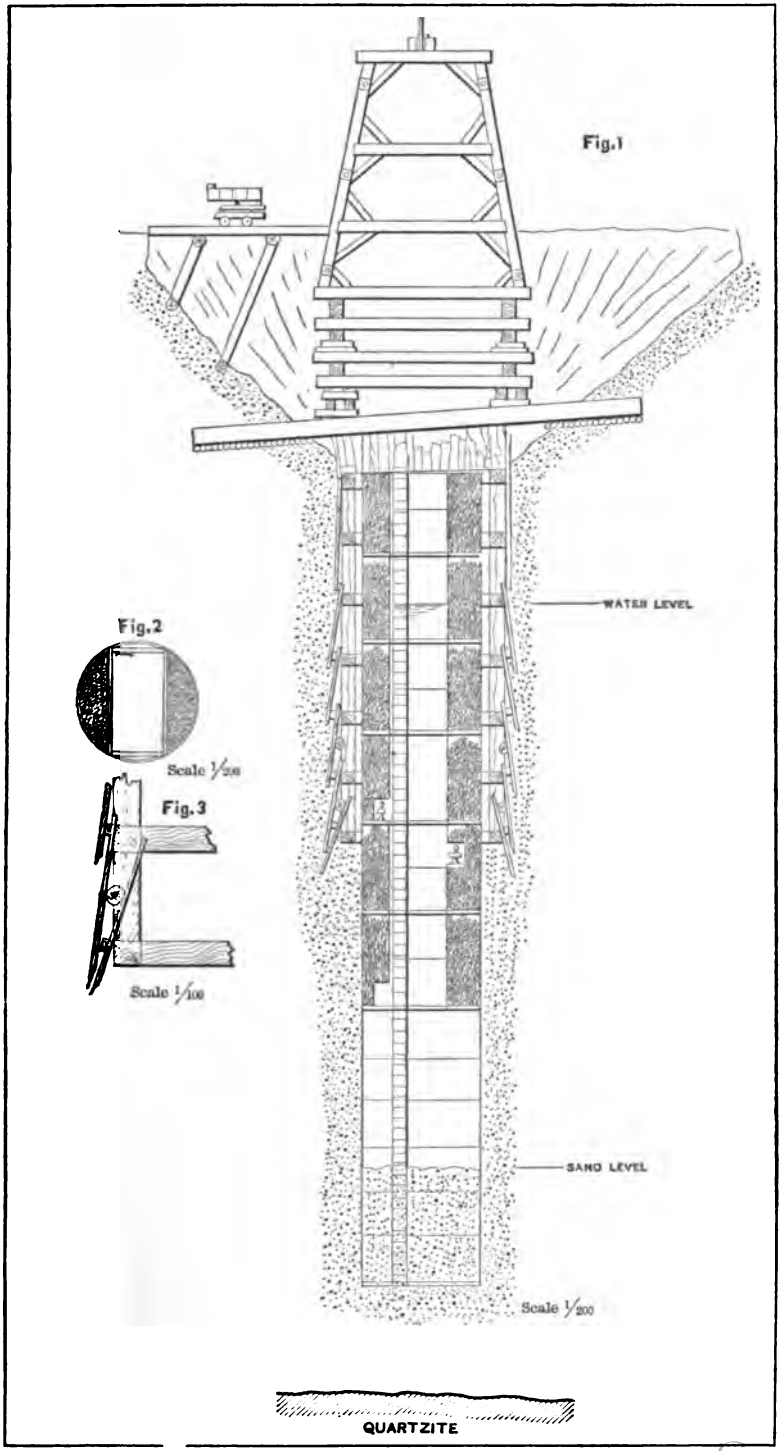

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pine blocks, 14 inches square and 4 feet long, set on end, one at each corner. This set being firmly placed, the sinking was continued until a depth was reached at which another set of timbers had to be put in, this set being suspended from the set above in exactly the same manner as before.

This continued until the first set of sheeting-boards had reached its depth, when the boards had to be driven from the inside and the regular fore-poling system was adopted. The poling-boards were made of hard wood and were about 7 feet long, 2 inches thick, and 8 inches wide, sharpened at the entering end. There was a set of round timbers, 9 inches in diameter, put in midway between every two sets of square, to give the poling-boards the necessary support and also to guide them; but, as they were not needed after the poling-boards had left them, they were taken out and used over again below. When the wedges between the lower ends of the poling-boards and the timbers were taken out, and a new set of boards started, there still remained a 3-inch by 6-inch plank between the boards to keep them apart. See Fig. 3 of plate.

This system worked very well, and by the end of February about 31 feet of sand had been taken out, but after sinking a little deeper, quicksand was met with, and with great difficulty the shaft was sunk about 10 feet lower, when the water and sand rushed in from below, and filled up the bottom of the shaft so quickly that the men just escaped with their lives.

When the water and sand rushed in at the bottom, the whole structure, shaft-house and all, commenced to sink with a terrible groaning and cracking noise, and it did not stop until it had sunk about five feet or more.

After a short time the damages were repaired, the pumps started, and the shaft was soon free from water; then the workmen went down and began to take out the sand that came in, but when they got down to about the same distance from the bottom as they were before, the sand and water came in again with the same result, the workmen having a narrow escape, and the structure sinking about the same distance as before.

Before work could be resumed the shaft-house had to be raised some, and timbers 12 inches square crossed under it; then the operation already described was repeated, with exactly the same results as before.

A thorough examination of the timbering was then made

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and it was found to be very much strained and twisted, and it was then decided to change the system of sinking.

The next plan adopted was to sink an iron caisson by weighting it and then pumping the water out and allowing it to rush in again, thus making room for it to drop down by the displacement of the sand. The larger part (48 feet in length) of the caisson was made during the month of August. It is 10 feet 6 inches in diameter inside, and consists of sections or rings 4 feet high, each section being made in three parts. The sheet-iron of which it is made is one-half of an inch thick, and each part has a three-inch flange all around its edge, so that it can be fastened securely to the parts on each side of it, and to the sections above and below. The flanges are braced by angle-irons three-sixteenths of an inch thick, and the parts forming the sections and the sections themselves, are bolted together by ¾-inch bolts, four inches apart between the centres. To keep it from leaking, pieces of ⅜-inch hemp rope, well tarred, were placed between the flanges before fastening them. A steel section, one foot high and in three parts, was fastened on the bottom by rivets. Its edge is quite sharp, and it projects about six inches below the sheet-iron.

When a number of sections of the caisson were ready, they were put together, one by one, directly over the shaft, and as the work proceeded the caisson was lowered, and when it stood on the bottom there was about two feet of the sixth section above the water; the remaining sections were then put on, making twelve in all, and the timbering commenced on the inside, beginning at the bottom and proceeding upward, the bottom timbers resting on the lower flange of the seventh section, which is twenty-four feet from the bottom of the caisson.

The timbering was so arranged as to leave a clear shaft with a cross-section of nearly 5 feet by 9 feet, and the spaces left on each side of this were closed in with thick plank and sills placed across, every eight feet. These cells were filled, as the timbering proceeded, with old iron, poor ore, and any heavy substance that could be found, care being taken to keep the weights on each side as nearly equal as possible.

The ladders were fastened vertically against the timbers at one side of the shaft, and a pump was placed in the shaft in a small space left for it in one of the cells, a little above the level of the water, and the discharge-pipe arranged so as to empty the water into a launder which carried it off to the swamp.

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When everything was ready the pump was started and the water all pumped out, and a number of men set to work to send up the sand, which they continued to do until they nearly reached the bottom of the caisson, at which point the sand and water rushed in, the caisson sinking as the sand was carried from under it, until it had settled about ten feet. The workmen, always being on the watch for this occurrence, escaped without much danger.

The pump was again started, the shaft pumped dry, and the excavation resumed; but the sand became more treacherous and the men had more difficulty working with it. Consequently, the results were not nearly so good, and the work progressed very slowly. It was, nevertheless, continued until a depth of 58 feet was reached when, after a long and hard struggle, a further drop of two feet was accomplished.

It was now discovered that the valves of the pumps were completely worn out by the action of the sand passing through them, and before proceeding a new pump had to be procured, which was ordered to be made very large and so constructed that it could pump sand mixed with water without injury to the valves. As the pump had to be made it caused a great delay, but as the men were needed to repair damages done to shaft A by a fire, they were kept at work in the meantime.

As the caisson stands now it is 71 feet long, new sections having been put on at different times. The last or top one is made of ¼-inch plate and is only 3 feet high, but it will have more than one foot of its length out of the water when the caisson rests on the ledge. The timbering has been carried up to the top and all the cells loaded. The total weight of the ballast is about 75 or 80 tons.

The top of the caisson is 21 feet below the grass level, and the bottom is only 10 feet from the ledge, but the sand is about 10 feet deep in the bottom of the caisson and the water is within 11 feet 6 inches of the top.

There are two pumps still in the caisson; one is a No. 5 Cameron, which is 18 feet, and the other is a No. 8 Knowles, which is 22 feet, below the surface of the water. The supply pipe, which is two inches in diameter, is not large enough and will have to be replaced by a larger one; but the discharge pipes, 4 inches in diameter, and the exhaust pipes, 2½ inches in diameter, are all in good order and can be used again.

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The large foundation-timbers have sunk down about 18 feet in the sand, and having sunk faster on one side than on the other, have consequently broken the iron rods on one side which held the timbers together, and made it necessary to block up the shaft-house so as to get it plumb again.

Arrangements had been completed for the repair of these damages and to have things ready to resume work when the new pump came. This was expected about the first part of August.

THE INCLINED PLANES OF THE MORRIS CANAL.[1]

By HERBERT M. WILSON, C.E. ('81).

THE difficulty of raising canal boats over great falls, requiring a series or flight of locks, considerable time, and great expenditure of water in the operation, led to the adoption of other means, viz.: (1) perpendicular shafts; (2) inclined lifts, or planes. The former, though used on the Great Western Canal, England, are not of a sufficiently extended application to require attention. The inclined lifts, however, have been, and are at all times, for falls of considerable height, the most economical. Like many other things, these lifts were first carried out by the Chinese. The first application, however, to modern canal systems is due to William Reynolds, who introduced them, in 1792, in the Shropshire Canal. Subsequently this system came into extended use on the canals of England.

THE MORRIS CANAL.

This canal was chartered December 31, 1824; began July, 1825, and completed from the Delaware River to Newark, during August, 1831, and extended to Jersey City in 1836. The planes and locks were enlarged in 1841. Its original dimensions were as follows: Canal—width at bottom, 20 feet; at top, 32 feet; depth of water, 4 feet. Locks—chambers, 9 feet wide by 75 feet long between mitre-sills. Planes—to correspond with locks, first constructed on various plans, 20 summit- and 3 lock-planes in all. The boats carried cargoes of 20 gross tons. During the winter of 1835–6, the summit-planes were altered to lock-planes. They were widened 2 feet, and the lock-

[1] Abstract of a memoir submitted by Mr. Wilson.

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chambers enlarged to 11 by 95 feet in 1841. The canal was sold in 1844, and the new company organized October 21, 1844. In 1845 the canal was enlarged, the width being 25 feet at the bottom, and 40 feet at the top, and the depth increased to 5 feet. The section boats were first introduced in 1845, and carried cargoes of 45 gross tons. From 1850 to 1860 all the planes were again altered to summit-planes, rebuilt and adapted to wire-rope haulage.

Elevations:

Mean tide-water to canal summit.

12 inclined planes	757 feet.	
16 lift locks	157 "	914 feet.

Delaware River (low water) to canal summit.

11 inclined planes	691 feet.	
7 lift locks	69 "	760 feet.
Total rise and fall		1,674 "

Summary of Cost in Round Numbers.

From Delaware River to Newark	$2,000,000
Alterations of planes in 1835–6	230,000
Extension to Jersey City in 1836	600,000
Greenwood reservoir and feeder	170,000
Enlarging planes and locks in 1841	400,000
Total	$3,400,000
Enlarging canal and rebuilding planes	1,700,000
Total cost	$5,100,000

The greatest rise in any plane is 100 feet, its length being 1,600 feet (near Washington, N. J.). The summit is at Port Morris, 41.34 miles from the Delaware River, and 60.80 miles from the Hudson River. The least rise of any plane is 44 feet. It is 3.4 miles from the Delaware, and 99.11 miles from the Hudson River. The whole length of the canal is 102.15 miles, the longest level (at Paterson) being 17.5 miles.

The boats, which are in two sections, are joined together by latches and steadying-pins, the ends abutting against each other. Transverse partitions of wood separate the two compartments, each of which is really a boat by itself. The aver-

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age tonnage is about 65 tons, but the planes can transfer boats of as much as 100 tons weight. The sectional system of boats was not adopted to suit the planes, but was previously introduced. It is, however, of great advantage in the use of the planes.

The first of the class of planes at present in use was introduced in 1848, at a cost, including machinery, of $27,835. It is 900 feet long, and the fall 51 feet. It superseded six locks.

The track of the plane in each case runs a short distance along the bottom of the lower bay, under water, rises up the incline to above the water level of the upper reach, then descends into the upper reach and runs a few feet along the bottom. The grade of the inclines is in general about 1 to 11.

The trucks which carry the boats, are, like the boats, divided into two sections, each section having eight wheels with flanges on each side of the rails. They are provided with strong stanchions to which the boats are fastened with hawsers.

The planes are in each case worked by a reaction water-wheel, and the levers for regulating the supply of water and for the control of the brakes are in a high tower, from which the man in charge can see the whole plane. This tower contains also the water-wheels and other machinery and is about midway between top and bottom of the plane and at the end of the flume.

The water-wheels have four arms and describe a circle 12 feet in diameter. The openings for efflux of water at the ends of the arms, are 15½ inches high by 3½ inches wide, and the wheel is placed far enough down the incline to get a head of 45 feet. The discharge is 1,000 cubic feet per minute, and 235 horse-power produced.

The quantity of water needed for these wheels is less than one-twentieth of the amount expended in a series of locks of the same total height or lift.

The first boat tried on the plane which was opened in 1848 was taken up in 3½ minutes, the weight of boat and cargo being 70 tons.

These inclines were constructed under the direction of Messrs. Asa Whitney and W. H. Talcot, chairman and engineer of the company.

The wire rope and the trucks used on these planes were manufactured by J. A. Roebling & Sons, of Trenton. The winding-drum is 12 feet in diameter, and is worked by the

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water-wheel; it has a continuous spiral groove of 3-inch pitch in its periphery. The rope is fastened at opposite sides of the drum, so that as one end winds the other unwinds. The motion is rendered reversible by a clutch on the jack-shaft of the water-wheel.

The Stanhope plane is of the same general type as all the planes west of the summit, and may be taken as an example.

The plane has a single track of two lines of heavy steel rails, 12 feet 4½ inches from centre to centre. The rails are 3⅛ inches broad at top, 3½ inches high, and weigh 76 pounds to the yard. They are spiked to longitudinal stringers of wood 6 inches high by 8 inches wide, resting at intervals on large flat stones two-thirds imbedded in the ground.

The car or cradle is in two sections, fastened together by a chain and a link. Each section is provided with snubbing posts, by which the boat is secured in the proper position as it floats into the car. Long "fender" boards on each side serve to support the boat when it is hauled from the water.

The wire cables are so arranged that as one winds on the drum the other unwinds. The two ropes pass around submerged horizontal sheaves at the bottom and top of the plane. The car has a wire rope attached at both ends, the "back-rope" to one section, and the main rope to the other. The latter is fastened to a small drum on the car, by which the slack can be taken up and the rope kept taut. Each section of the car has eight double-flanged wheels, provided with brakes.

If the car is to be drawn out of the lower reach and up the plane into the upper bay, all that is necessary is for the engineer in the plane-house, called the "plane-man," to turn the "tub-wheel" which lets the water into the reaction water-wheel, and the drum winds up the cable at one end and unwinds it at the other, drawing the car up.

To take a boat down the plane, if it is empty, it is hauled out of the upper reach, the water shut off the wheel, and the car allowed to descend by its own weight. A boy on the car can apply the brake if the speed of descent becomes too great. If the boat is loaded, the plane-man puts on about half water—that is, opens the tub sufficiently to allow one-half the amount of water for full power of wheel. This prevents the boat from going down too fast. The planes west of the summit are uniform; those east of it vary somewhat. At Drakesville, for instance, the plane is 1,770 feet long from centre of

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wheel-pit to centre of wheel-pit, its total rise is 50 feet grade—1 in 10, it requires 3,900 feet of cable to work it, and the total head on the wheel is 30 feet. It differs mainly from the Stanhope plane in having but two lines of cable instead of four, and but two grooved pulleys. This simplifies the construction materially, and makes a great saving in wire cable, pulleys, pulley-block stands, etc. Besides, the plane works more easily, and there is less slack. Instead of passing out in the same direction from opposite ends of the perpendicular diameter of the drum, it passes out in opposite directions from the same end of the diameter. Instead of being carried all the way on small pulleys, the cable is supported near the water's-edge on two large vertical 11- or 14-foot groove-wheels. These wheels are in large masonry pits or slots in the ground, so that their upper surface is but a little above the surface of the ground.

All of the east side plane-houses are two stories high above the ground instead of three stories, as on the west side, the brake and reversing-lever attachments being thereby greatly simplified.

All of the water-wheels are covered with a plate of iron, above and below; this entirely covers them, excepting a few inches over the nozzle. In all other respects these planes are entirely similar to the one at Stanhope.

At Washington and at Newark there are planes of a different construction. These are double-tracked, two double lines of rails running parallel and the whole length of the plane. There are two cars, one ascending while the other descends, meeting half-way. The cable is arranged as at Stanhope. This arrangement relieves the machinery of part of its work, as the descending car helps in raising the other one.

From careful observation, I find that to take a loaded boat up the plane at Stanhope, from the time it starts below until it just floats in the upper bay, it takes from 5 minutes 10 seconds to 6 minutes. The average being about 5 minutes 30 seconds. For lowering a loaded boat, on the average about 2 minutes 40 seconds are required; for an empty boat, 2 minutes 50 seconds. For an empty car, without boat, 2 minutes 45 seconds. As it would take about four such planes in length to make a mile, it would require 11 minutes to draw an empty boat a mile up such a plane. For a descending loaded boat, 9 minutes; for an ascending loaded boat, 22 minutes. These figures are as near as can be approximately reckoned,

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and equal the ordinary rate of travel of the boats when drawn by mules, about one mile in 30 minutes loaded and one mile in 20 minutes unloaded. Hence we see that, unlike the locks, the boats are being raised and at the same time proceed at their ordinary rate of travel; for, although while on the plane the speed is somewhat greater than in the canal, allowance must be made for the few minutes spent in getting the boats into the car; besides, in going a mile, the boat rises vertically about 300 feet on this particular plane.

From the above we find that while a boat takes probably about 8 minutes to go through a lock of 6 feet rise, to go through a flight of 12 locks, equal to a plane with a rise of 70 feet, would take 96 minutes; and during all this time a boat not only in passing a plane loses nothing in horizontal motion, but by the saving of time is enabled to advance about five miles while the other boat is passing the locks. The saving of time is evidently considerable.

On the whole canal there are twenty-three planes, with an average lift and length of that at Stanhope—the total length six miles. It takes the empty boats 66 minutes and loaded boats 198 minutes to travel this distance, and as there are as many boats going down as there are going up, the average time consumed in travelling these six miles is 133 minutes or one mile in 22 minutes, which is better than ordinary canal speed.

If, instead of these planes, there were twenty-three flights of locks, each one consuming 96 minutes in its passage, the whole would require a loss of 36 hours, or in distance—at the rate of one mile in 22 minutes—of 100 miles!

The cargoes carried on this canal are almost exclusively coal and ore, with occasionally a load of grain or wood. Of wood, grain, or coal, the boats take a full load to sink to the water-line, but ore being heavier for the same bulk, a very little in the bottom is all they can carry. The usual load is 70 tons, but sometimes 75 and 80 tons are carried; the latter, however, is uncommon. The empty boats weigh from 14 to 18 tons, average 17 tons. The cars alone weigh from 38 to 45 tons, and average 40 tons; hence, the average weight raised on the planes is 127 tons, and it may be as high as 143 tons.

The grades are never very steep; at Stanhope the grade is about 1 in 10; at Port Morris it is 1 in 20; the steepest, 1 in 9; average grade, 1 in 11.

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Expenses.—The first cost of a plane considerably exceeds that of a single lock, as do also the running expense, repairs, etc. A plane with a rise of 70 feet, however, will cost very nearly the same as a flight of six locks of a rise of 12 feet each.

The wire cable costs about $1 per foot, and needs replacing about once in three years. The large drum costs about $3,000, and lasts many years. The entire machinery needs replacing about once in ten years, with the exception of the drum and shafts, which last much longer. In locks there is very little repairing to be done, with the exception of the wickets, which do not last, but are small and cheap.

We will compare an average plane, as that at Stanhope, with a lift of 72 feet, with a flight of twelve locks lifting each 6 feet. For a loaded boat, the plane takes 5 minutes 30 seconds = 330 seconds for passage. The water in the flume lowers 7 inches and flows at a velocity of 120 feet in 60 seconds = 2 feet in 1 second. The flume is 8 feet wide; hence, the wheel consumes $8' \times 2' = 16$ square feet of water per second, this multiplied by $7''$ in depth, gives the consumption of $9\frac{1}{3}$ cubic feet per second; and if the boat takes 330 seconds for its passage, the total amount of water required to raise a loaded boat from the lower to the upper bay is $330 \times 9\frac{1}{3}$ cubic feet = 3,180 cubic feet. To take a boat down, the water lowers 1 inch, the velocity is 48 feet in 60 seconds, equal to about $\frac{3}{4}$ feet in 1 second; this multiplied by 8 feet, the width of the flume, gives 6 square feet per second $\times$ $1''$ (the depth) = $\frac{1}{2}$ cubic foot per second; and for 3 minutes = 180 seconds it takes 90 cubic feet of water, which is expended in holding the boat back.

In a flight of locks, each 95 feet long by 11 wide, with 6 feet rise, we have 95 $\times$ 11 feet $\times$ 6 feet = 6,270 cubic feet for only one lock; for twelve such locks, equal to a rise of 72 feet, the amount of water would be = 72,240 cubic feet. Hence, the locks expend about 23 times more water than the planes for a loaded boat, and 836 times more for an empty one. This item of economy of water is of prime importance in canals, especially in dry seasons.

The question whether the locks or inclines can be most advantageously used on a canal for effecting a change of level is not readily answerable. The advisability of adopting one system or the other depends, in each case, on the supply of water obtainable and on the amount of traffic. The ex-

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pense is reduced by transferring a greater amount of load at one time; this requires a large expenditure in construction, and the advisability of making such expenditure depends on the amount of traffic. The planes, however, cost very little more than a flight of locks of the same lift, consume less than one-twentieth of the amount of water required by locks, and save 60 per cent. in the time of passage, as the average rate of travel (four to five miles an hour) is continued horizontally while the car ascends the slope.

To sum up, one lock is more economical than a short plane; a plane is more economical than a flight or series of locks, especially in the items of water and time. A plane involves more machinery, details, etc., than a lock, but not so much as to make it more expensive than five or six locks in series.

DETERMINATION OF "TOTAL CARBON" IN STEEL AND PIG-IRON.

By H. F. STARR, Ph.B. ('79).

No claim is made for originality, in any particular, for this method. All that is intended to be shown is how the determination may be made with speed and accuracy.

A concentrated solution of $CuCl_2$, as near neutral as possible, should be made, diluting this to three times its volume. Also a concentrated solution of $CuCl_2$, in which the presence of hydrochloric acid makes no difference.

For steel, weigh out 3 gr., and for pig-iron, 1 gr. Put into a small beaker holding about 250 c.c., and add 50 c.c. dilute $CuCl_2$ solution. For a few moments the contents of the beaker must be kept in motion, in order that the copper may come down *spongy;* if it is allowed to stand, the copper will form crystals exceedingly hard to dissolve. The reaction of the $CuCl_2$ on the steel produces considerable heat, and it is necessary that, to prevent loss, it should be held in cold water occasionally.

With soft steel or pig-iron the operation may take a little longer, but in the case of hard steel, especially when the shavings are thin, five to ten minutes are long enough to insure the complete precipitation of the copper. Add 50 to 75 c.c. concentrated solution of $CuCl_2$, and 10 to 15 c.c. concentrated

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hydrochloric acid; place on a water-bath, and when thoroughly heated, commence stirring at short intervals, until complete solution of the copper is effected, then filter. The funnel should be made of soft glass tubing, 4 or 5 in. long, with an inside diameter of ¾ in., drawn to a point, and broken off. Put in pieces of broken glass up to the shoulder, and then a layer of asbestos. The way in which the asbestos is put in determines largely the length of time required for the analysis. If pressed in too hard, especially with very soft steel, it may take a whole day to filter. It can readily be arranged to allow the liquid to run through almost as fast as it can be poured in. After the solution has been filtered, add 4 or 5 c.c. hydrochloric acid, to insure complete solution of cuprous chloride. Wash well with hot water, using at least 200 c.c.; then add 10 c.c. absolute alcohol, and draw this off as completely as possible, either with the aid of a filter-pump, or by attaching a piece of rubber tubing to the end of the funnel and drawing with the mouth. Dry in an air-bath, and then push out carbon and asbestos into a platinum boat. Cleanse the tube thoroughly with a little dry asbestos.

In my own work I use a platinum combustion-tube with platinum caps fitting over the ends of the tube. *One* U-tube filled with fine calcium chloride, between the combustion-tube and the U-tube containing the soda lime, is sufficient; the more drying tubes there are, the greater chance for loss.

Perhaps the most simple way to fix a soda-lime tube is to put, midway between the top and bottom of that side of the tube farthest from the combustion-tube, a bit of cotton; on that a half inch of granulated calcium chloride, and then another bit of cotton. The rest of the tube is of course to be filled with soda-lime. I prefer soda-lime to a solution of caustic potash for two reasons: first, by using soda-lime you can always tell when the combustion is completed by the cooling down of the tube to the temperature of the room, and second, there is no resistance to the gas passing through.

Always run oxygen gas into the soda-lime tube before weighing. I never use oxide of copper to help the combustion, but simply employ oxygen both to burn the carbon, and instead of an aspirator to cool it.

The whole operation may readily be completed in two hours.

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PROFESSIONAL NOTES.

Mining Engineering.

The following sections of two shafts sunk under the direction of Thos. F. Van Wagenen ('70) may prove of interest to some of our readers:

Geological Section, Shaft No. 1, Tip Top Mine, East Fryer Hill, Leadville, Col.

	Feet.
Surface debris, having the appearance of a morainal deposit	95
Hard gray trachyte	140
Black shale, greasy and brittle, with iron pyrites slightly disseminated.	5
Hard white trachyte	38
Black shale (as before)	10
Soft yellow trachyte, fracture planes stained with iron and manganese oxides carrying trace of silver. Occasionally fragments of chert (angular) containing bunches and vugs of iron pyrites assaying 5 to 20 ounces silver per ton	8
Coarse blue sand, composed of disintegrated blue limestone and trachyte assaying slightly of silver	30
Rounded fragments and masses of brown and black silicious hematite showing silver chlorides; spaces between filled with decomposed trachyte	10
	336

At this depth the flow of water overcame the pumps, and compelled suspension of work until more powerful machinery could be erected.

Geological Section, Shaft No. 2, Amie Mine, Fryer Hill, Leadville, Col.

	Feet.
Morainal wash and debris	80
Very hard gray trachyte	20
Brown hematite containing bodies of lead carbonate rich in silver. This layer has produced up to date about $1,200,000	20
Trachyte, same as above	50
Hematite as above, but containing only small and unprofitable bodies of ore	5
White and very hard trachyte	70
Hematite similar to last layer	5
Soft yellowish trachyte, fractured into small blocks	45
Wash, consisting of a mass of decomposed trachyte containing rounded boulders of limestone, quartzite, granite and very silicious trachyte	50
Trachyte hard	20
Quartzite	110
	475

The quartzite for the lowest 100 ft. was very hard and dense and white.

Cement as a Lining for Mine Gangways.—A novelty in the way of protecting headings and as a partial substitute for timbering, has been introduced on a large scale in some parts of Austria. Herr W. Stieber has given an account of it before a local society. Even in good rock, gangways and headings will require timbering after awhile, the variations of temperature and the water acting in addition to other deteriorating causes; and prices of timber and cement being favorable, 13,000 lineal feet of heading have thus been coated in the Wiczek mines, over 3,300 feet having been put in four years ago. The best way is, of course, to use the cement in the shape of masonry. Herr Stieber has, however, simply coated his gangways with a layer of cement ranging

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in thickness from 0.3 to 0.4 inch. The strata of the rock must not be so weak as to break down within a week, as the drying of the cement often requires from three to four days. The thickness varies according to the nature of local circumstances. The surface of the heading is made as even as possible, wet with water, and then the cement is put on with a trowel, all fissures and cracks being filled up carefully. When partially dried, new layers are put on until the right thickness has been obtained. The surface is smoothed and is sprinkled with water from time to time during the next few days, in order to prevent cracking by too rapid drying. Both at the Wiczek mines and at the Johann-Marie, at Polnisch-Ostran, this cement lining proved cheaper than timbering. The editor of the *Oesterreichische Zeitschrift* states that in a limited way this method had been employed in Prussia as early as 1871, and we may add that it is carried out on a large scale by Gen. William Sooy Smith and son in lining the Hudson River tunnel.—*Engineering and Mining Journal.*

The method has been used since 1871 at the Piesberg Mine, Westphalia. The shales over the coal, under the influence of water and air, shelled off badly and gave much trouble and expense in keeping the gangways clear. By the application of a thin coating of cement, requiring $\frac{1}{16}$ ton cement per lineal yard of gangway and involving a total cost of about 50 cents per square yard, the trouble was obviated, saving $30 to $40 per year in the expense of keeping the roads in order.

Brickwork Dam in the Dorchester Sewage Tunnel.—A paper on "Brickwork under Water-pressure" was read by Mr. D. McN. Stauffer before the Engineers' Club of Philadelphia, April 1st, 1882. During the construction of the tunnel under Dorchester Bay in Boston, it became necessary to construct a brick bulkhead that had to withstand the pressure due a water column 162 ft. high, and as it was intended to shut off a leak amounting to 240,000 gallons per 24 hours, it had to be water-tight. The tunnel section at the point selected for the bulkhead was practically 10 ft. square. The bulkhead was built directly across the tunnel, 50 ft. in front of the heading where the water was struck. Plank dams filled with puddle clay were first thrown across the tunnel, each side of the bulkhead site, and a 6-in. wrought-iron pipe used to carry off the water during construction—the pipe being built into the brickwork. An arched form was adopted, 4 ft. thick at the crown and 2 ft. rise in a span of 10 ft. Hard, burned bricks were used, laid in mortar made of one part English Portland, one part Newark Rosendale, and two parts clean, sharp sand—a compound found equally as strong as English Portland, and sand one to one, and having the advantage of working smoother on the trowel, and adhering better to brickwork in wet places. Skewbacks for the arch were roughly picked out in the rock at the sides. After the cement had set, the water was shut off by screwing a cap on to the outer end of the 6-in. pipe. The pressure against the wall was $72\frac{1}{6}$ lbs. per square inch, or about 519 tons distributed over the face of the bulkhead. The wall was tight for about 48 hours; then the water came through the *brick* itself, rather than through the joints, in amount equal to one-half the original volume.

The water was let off and a second experiment tried. The main wall was torn down sufficiently to allow men to pass behind it, and a

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second wall only 12 in. thick was built back of the first, and 2 ft. distant. The space between these walls was well rammed with puddle clay, extending to the rock on all sides. The second wall was made purposely light, as its yielding to the pressure would only more effectually consolidate the clay between it and the unyielding wall in front. The bond used in the main wall was one so laid that there were no continuous horizontal joints through the wall. The result of the last construction was to completely shut off the water from this leak, at a great saving of fuel and pumps. The Dorchester Bay tunnel is 7,000 ft. long, with its average invert grade 143 ft., below low-tide mark in the bay under which it passes. The tunnel is driven through a formation of clay, slates and conglomerates for its entire length; rock, very seamy, and much water encountered.

Topographical Models.—A paper by Mr. O. B. Harden, of the Pennsylvania Geological Survey, was read at the last meeting of the Engineers' Club of Philadelphia, in which the method pursued by the survey was explained, and the text illustrated by photographs and models. The requisite for a good model is a good contour map of the ground to be illustrated, with contours 10, 20 or 100 feet vertically apart. The lumber used by cigar-box makers is best suited for modelling—of a thickness required by the vertical scale. On a scale 1,600 feet to an inch the boards will be $\frac{1}{16}$ of an inch thick and represent 100 feet in height. The contour line marked upon the base and the one next above it are traced upon boards cut the exact size of the base, and the lowest or outer one of the two is sawed through by a jig-saw, the upper or inner one serving to mark the place for the next succeeding board. These are nailed on with sprigs—glue being used where, from the sharpness of a piece, a nail would split the wood. The wooden model is then waxed, which is so made as to be quite plastic by the warmth of the hand, the wax[1] being worked down until the edges of the boards are just seen, and until the model presents a smooth appearance and fac-simile of the topography. The surface is rubbed with linseed-oil and is then ready for casting in plaster of Paris. Care should be taken to get the oil in on the low places, as on this depends the facility with which the negative can be removed after the setting of the plaster. A cast is then made with plaster of Paris, filled in around a frame to the requisite thickness, and, after being allowed to dry for about an hour, it is removed from the original, smoothed up with a file or knife edge, and coated with shellac varnish. From this cast a positive is made—pursuing the same method as before—and is dried and scraped. The sides are smoothed up for the sections, and after being varnished the model is ready for the coloring—representing the different geological formations. Streams and geographical features being added, the model is complete.

Transportation of Coal on the Ohio and Mississippi Rivers.—In a recent number of *Harper's Monthly*, is an article, illustrated by numerous engravings, describing the transportation of coal in flat-boats down the Ohio and Mississippi. Over 3½ million tons of coal were shipped from Pittsburg during 1880, the bulk of which went to Cin-

[1] 16 parts yellow wax, 8 parts corn starch, 1 part Venetian red, 4 parts Venice turpentine, and 1 part sweet oil.

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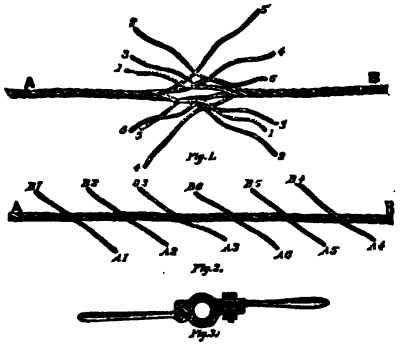
cinnati, Louisville, Memphis, Vicksburg and New Orleans. For long distances, flat-boats of two kinds are commonly used. The first, called "boats" simply, are of slight construction, built of 1½-in. plank, 180 ft. long, 26 ft. wide and 9 ft. deep, holding 840 tons, and costing $700 each. A more substantial form, the so-called "barge." are of somewhat smaller dimensions, 160 to 180 ft. long, 24 ft. wide and 7½ ft. deep, drawing 6½ ft. water, holding 530 tons, and costing $1,100 each.

A large number of boats are lashed together in a single tow, like an immense raft, and *pushed* down the river by powerful steamboats, 2,000 H. P., often as much as 20,000 tons in a single "tow," or enough to load five ocean steamers. A speed of about nine miles per hour is attained, and the cost of transportation from Pittsburg to New Orleans, 2,000 miles, averages $1.30 per ton, or $\frac{6}{100}$ cents per ton per mile.

Splicing Wire Rope.—The increasing use of endless wire rope for underground haulage, elevated wire ropeways, and for hoisting, gives importance to methods of splicing. The following is taken from Hallidie's pamphlet on "Wire Ropeways":

About eighty-four feet of rope is required to put in a good smooth long slice. The wire ropes employed in these ropeways are made six strands of seven wires each, and a core or heart; as there are two rope ends to splice together, there will consequently be twelve strands to be tucked in. Operators usually tie the stops that mark the length of rope, about where the centre of the splice will be. In this case the usual way is to unlay each rope up to that point, and place the strands of rope A between the strands of rope B, the core or hearts of the ropes A and B being cut off so that the cores of the ropes abut against each other. There will be then forty-two feet of strands each side of the stop, as is shown in Fig. 1.



It is important that each strand should be in its proper place, so that none of them cross other strands, or that two strands be not where one strand should be (by placing your fingers between each other in natural position, this will be understood). Then strand No. 1 of rope A is unlaid, and strand No. 1 of rope B follows close, and is laid snugly and tightly without kink or bend in its place, until within seven feet of the end; a temporary seizing is then put on securing ropes and strands at this point. Strand No. 1 of rope A is then cut off, leaving it seven feet long. Then strand No. 2 of rope A is unlaid, and strand 2 of rope B is laid in its place to within twenty-one feet of its end. Strand No. 3 of rope A is unlaid, and strand No. 3 of rope B is laid in its place, within thirty-five feet of end. By this time you have reached within seven feet of the centre, and reversing the operation, unlay strand No. 4 of rope B, and lay in its place strand No. 4 of rope A, to within seven feet of its end; unlay No. 5 of rope B,

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and lay in No. 5 of rope A, to within twenty-one feet of its end; finally, unlay No. 6 of rope B, and lay in its place No. 6 of rope A, to within thirty-five feet of its end. The strands are now all laid in their places and seized down for the time being, the ends are cut off, as with the first strand, to seven feet in length, and present the appearance as in Fig. 2.

The next operation is to tuck in the ends, and we will proceed to tuck in B 1. It will be remembered that the ropes are made of six strands, laid around a core or heart, usually of hemp, of the same size. Two clamps (Fig. 3) made for this purpose are fastened on the rope so as to enable the operator to untwist the rope sufficiently to open the strands and permit the core to be taken out (see diagram), which is cut away, leaving a space in the centre of the rope; the strand B 1 is placed across A 1, and put in the centre of the rope in place of the extracted core, forming in fact a new core. A flat-nosed T-shaped needle used in splicing, the point of which is about one-half inch wide by three-sixteenths of an inch thick, rounded off to an edge, is well adapted to this purpose. The strand B 1 is laid in its entire length, the core being cut off exactly at the extremity of strand B 1, so that when the rope is enclosed around the inserted strand, the ends of the strand and core should abut. If there is much space left in the centre of the rope without a core, the rope is liable to lose its proper form and some of the strands fall in, exposing the projecting strands to undue wear. The same operation is performed with A 1, running the other way of the rope, and so on, until all the strands are tucked in, which, if properly done, will leave the rope as true and round and as strong as any other part.

Other operators prefer to start from the end of one rope and consequent end of splice. The operation is about the same, but the experience of the writer justifies him in saying that more care has to be used in bringing all the strands to an even tension in the parts spliced. Other variations in detail are made according to the fancy or practice of the splicer, but after making a few successful splices in manner above described, the operator can afterward vary to suit himself.

Nobel's Blasting-Gelatine and Gelatine-Dynamite have proved very efficient in the St. Gothard Tunnel. The principal fact established during the use of these explosives is that the relation between the blasting-gelatine and Nobel's common dynamite is about 3 to 5.

Gelatine-dynamite has more of a pushing than a breaking effect, and hence advantageously applicable to quarries, and, on the whole, its effect and economic value is about the same as that of common dynamite. With the stronger of the new explosives, blasting-gelatine, where boring is expensive, the same work may be accomplished with the boring of 3 holes, as with 5 holes charged with common dynamite. The gases from both gelatine-dynamite and blasting-gelatine did not inconvenience the workmen, even immediately after the blast.

Ore-Dressing.

Crushing.—Stamps with cylindrical heads have been introduced at Meinerzhagene-Bleiberg, near Mechernich, in the Bonn mining district, and have proved satisfactory. Two stamp-mills, of 115 and 160 heads respectively, are in operation. Chilled steel is found best for

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stamp-heads and for the shells of crushing rolls. At the Clausthal dressing-works, steel has replaced cast-iron for crusher plates and mortar bottoms.

Spiral Sieve.—At the Beilstein iron mine, Dillenburg district, the experiments with this form of sieve have proved satisfactory, one of these having replaced three separating drums with great economy of power, space, and water. Repairs more rapid and cheaper than with drums heretofore used.

Feeding Apparatus for Round Tables, Meinerzhagene-Bleiberg.—In the mill and stamp-house a feeding apparatus is used, consisting of a four-cornered box, with a sieve 5–10 c.m. above bucket-shaped bottom, fine enough to keep dry sand from passing through. Below this is a tube with rose-jet, by which water is forced into the sand from below. Water regulated by stop-cock, and thus the flow of sand. A second tube below adds to the slime the water necessary for the washing.

Saving of Mineral in Waste Slimes.—At the same iron mine, the slimes are lifted 13.5 m. above the floor of dressing-works by means of sand-pumps, by which the sands are spread in a broad stream on the dump. Waters allowed to clear in settling-pits. In the pipes an apparatus is placed which separates the quartz containing grains of mineral. The pipes have a rise of 6° and a clear width 40 c.m. on 80 gas-pipes with stop-cocks below, 1–3 m. long and 12 mm. clear diameter, by which the rich sands are drawn off as they accumulate. The lengths of the pipes correspond to different pressures, so that by friction in them the semifluid mass may drip slowly out of the tubes. Out of 2,000 tons of sand which go through main pipes per twenty-four hours, a concentrate is obtained enriched to 6 and 12 per cent. which after further dressing produces a 30, 40, or even 50 per cent. ore.

At the Clausthal works the settling-pit system has been enlarged by the addition of larger pits in which fine slimes are caught, heretofore lost. An experiment on a large scale showed that 1 c.m. of slime produced 22.33 kilos *schlich* having value of $1.10 with an expense of 12 cents for washing. By this means a saving of $918 would be obtained from 900 c.m. of settled slime per year.

An enlargement of slime-pits would generally therefore seem to be desirable, and especially as tables can generally wash this material without enlargement of plant.

Magnetic Dressing at the lead and silver Friedrichssegen Mine, Oberlahnstein.—Blende mixed with carbonate iron gangue has for some time been separated by magnetic dressing—ore calcined, carbonate converted with magnetic oxide, and separated by magnets. Roasted in furnace. Time for schlich, one hour; and for pea sizes, one and one-half hours. Temperature of roasting, 600° C.

Civil Engineering.

Expansion of Iron Tie-Rods in Brickwork.—At a meeting of the Engineers' Club of Philadelphia, March 4, 1882, the Secretary presented, on behalf of Mr. Thomas U. Walter, honorary member, a description of his experiments in 1836–37, upon the effect of extreme

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temperatures upon iron bands imbedded in the walls of Girard College, Philadelphia, of which he was the architect. These bands, each 54 ft. long, were introduced to resist the lateral pressure of the arches. Although confident that no evil could result therefrom, Mr. Walter considered the positive evidence of these experiments both satisfactory and interesting. The south vestibule wall, 5′ 5″ thick, subjected to the full power of the sun during summer and in winter under temporary roof, with no fires in furnaces, was selected for experiment. Self-registering thermometers, minimum for winter and maximum for summer experiment, were placed upon the band and the whole walled in as finished work. The following changes, in Fahrenheit degrees, took place:

Sept. 23, 1836. }	82°.	{ Intermediate extreme cold, —3°.
July 29, 1837. }		{ Minimum temperature in middle of wall, 42°.
Jan. 16, 1837. }	24°.	{ Intermediate extreme heat, 94°.
Dec. 16, 1837. }		{ Maximum temperature in middle of wall, 61°.

Difference of temperature outside of wall, 97°. Difference of temperature in middle of wall, 19°.

Smeaton's co-efficient of expansion for malleable iron, $\frac{1}{794}$ of length for 180°, being somewhat greater than that of other experimenters, was used. The co-efficient for brickwork, $\frac{1}{1818}$ of length for 180°, was taken from experiments by A. J. Adie, C.E. (Royal Society of Edinburgh, April 20, 1835). The difference occasioned by a change of 19° in temperature in a length of 54′ is, therefore, less than $\frac{1}{50}$″, but if we consider that the exterior of the wall is subjected to the extremes of heat and cold, it will be obvious that the aggregate change of the brickwork is even greater than that of the iron.

Grouting with Hydrostatic Pressure.—In *Engineering*, February 24, 1882, the following novel method of using a grout of hydraulic cement for checking leakage in a graving dock at Greenock is described: The dock was built in 1785, is 223 ft. in length, 50 ft. in breadth, and has an entrance 34 ft. wide, with 10 ft. of water on the sill at high tides. It was in a very leaky condition, and it seemed necessary either to reconstruct it or to remove it entirely. Instead, however, of doing either, holes 3 in. in diameter, and from 1 to 2 ft. apart, were bored through the masonry behind the heel-posts down into the sand for several feet below the foundations. Similar bore-holes were also put down in the outer and inner aprons, a few feet from the pointing-sill. Stand-pipes with hoppers were then erected in these holes, and a thick grout of neat Portland cement poured down. The pressure due to the weight of the cement was sufficient to force it into all the fissures and cracks, forming a water-tight sheeting of Portland cement. The level of the water in the stand-pipes was 6 ft. below its level without. Five tons of cement were used, and the leakage has been reduced to $\frac{1}{24}$ of what it formerly was.

In *Engineering*, March 3, 1882, two similar operations are described. In 1879 a large sea-sluice was erected at the outfall of the River Glen. The foundations were in quicksand, and the floors and aprons were made of 3½-in. planks, secured to the timbers by spikes and treenails. Under the timbers there was 3 ft. of concrete. After the planks were laid it was found that the shocks due to the driving of the spikes had prevented contact between the planks and the concrete.

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This allowed water to leak into the sluice. Holes were bored into the planking, and stand-pipes 8 ft. high were set up and Portland-cement grout poured in. A small force-pump was also employed. The grout diffused under the planks, and the floors were made absolutely water-tight.

In another sea-sluice, at the River Nene, the foundations were in quick-silt. The sill was 6 ft. below low water, and the tidal rise was 20 ft. against the earth dam. At high tide the sluice leaked badly. To remedy this the old planking of the aprons was taken up and 3 ft. of concrete got in up to within about $\frac{1}{4}$ in. of the horizontal timbers. The concrete set well, but water constantly percolated through it. The $3\frac{3}{4}$-in. planks were laid within $\frac{1}{4}$ inch of the concrete. Portland-cement grout was then forced under the planks for thirty days. One leak after another choked up, until finally the floors were absolutely water-tight.

Under the direction of Mr. L. B. Emerson the same plan was tried with great success at the Copper Falls Mine, Lake Superior, for repairing the foundations of one of the huge steam-stamps. Holes were bored through the foundation and connected by 2-in. gas-pipe with a box on one of the upper floors of the mill. After the cement had been mixed to the desired consistency the plug was taken out, and the contents of the box speedily found their way through the pipe and into the foundations. By repeating this process at several points the foundation was made perfectly solid, showing not the slightest movement under the heavy blows of the stamp, while before this treatment a very decided jar could be seen with every stroke. G. A. S.

Flow of Water in Pipes.—An interesting study on this subject has been recently published by Albert Frank, of München, in the *Civil Engineer*, 1881, pp. 161–222. He first gives a brief historical account of the subject from Couplet (1732) to Zampe (1873), and the late experiments of O. Iben, who maintained Darcy's formulas for lost head

$$p = \left(0.001014 + \frac{0.00002588}{d}\right)\frac{v^2 l}{d}$$

for clear flow. The author proposes the following equation, as being more in accordance with his experiments:

$$p = \left(0.000512 + \frac{0.0003847}{\sqrt{d}}\right)\frac{v^2 l}{d}$$

It is only when the diameter exceeds 0.5 m. that the formula fails; for such the formula becomes

$$p = \left(0.000912 + \frac{0.00008}{d}\right)\frac{v^2 l}{d}$$

For pipes without any appreciable elimination of cross-section the author deducts the following from Darcy's experiments:

$$p = \left(0.000495 + \frac{0.000652}{\sqrt{d}}\right)\frac{v^2 l}{d}$$

d being the actual internal diameter.

Experiments on the Strength of Brickwork have been conducted by Professor Böhme at the Royal Testing Institute at Berlin. We have room but for the following important result: The compressive

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strength of masonry after three months is less than that of the individual bricks, and on the average as in the following table. Masonry with mortar as follows:

1 part of lime and 2 parts of sand.....	44	per cent.	of strength of brick.	
7 parts of lime, 1 part of cement, and 6 parts of sand.....	48	"	"	"
1 part of cement to 6 parts of sand....	55	"	"	"
1 " " " 3 " "	63	"	"	"

The working with a factor of safety of 10 would be 9 kilogrammes for common brick masonry in lime mortar; 14 kilogrammes for the same in cement mortar, and 20 kilogrammes for the best brick in cement mortar.

Blast Furnace Slag mixed with Limestone in the proportion of about 4 to 7 has been found to produce a very high grade mortar. Experiments show that after three days this mortar was harder than Portland cement, in seven days it was as hard as the cement after three months, and in 28 days it was harder than the cement after seven years.

The Removal of Snow in St. Petersburg.—The snow is thrown into pits, which are located at convenient points of the city. It is melted in these by steam, and runs off into the river by suitable channels.

The Phosphorescent Paint recently introduced in this country has been applied to buoys in England with considerable success, these being discernible at several hundred metres distance. It has also been applied on the Southeastern R.R. for coating the inside of railway cars. The light given was brilliant but mild blue in color, and was sufficiently strong to enable the reading of a watch. The principal objection is the high cost.

A Lighthouse on the coast of France, near Brest, has been built under conditions of exceptional difficulty on a rock level with the sea. Owing to the surf and current it was only possible to land seven times and work eight hours during a whole year. During the second year eighteen hours' work were put in, and it was only in the third year that a sufficient number of holes had been bored in the rock to permit the anchorage of 75 c.m. of masonry. The diameter of the tower at the base is 7.2 m., and it will be 30 m. high. The masonry is tied by iron bars sunk in the rock 1 m. apart.

Cost of Work with Various Motors.—Cost of one H.P per hour, as follows, from experiments lately made at Carlsruhe:

		Cents.
100 h.p.	steam-engine.....	1.90
2 "	"	11.07
2 "	Lehman caloric-engine	6.62
2 "	Hock motor.....	2.00
2 "	Otto gas-engine	6.52
2 "	Otto-Langen gas-engine	6.52
2 "	Schmidt water-engine (fed by city water supply).....	23.75
	Horses	11.10
	Men	50.00

K. N.

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Analytical and Applied Chemistry.

The Roughness on Platinum produced by a gas flame was found by A. Remout (*Bull. Soc. Chim.* [2], xxxv., p. 486) in one case to contain nearly half its weight of platinum—0.022 grm. gave 0.01 grm. The investigator considers that this is caused by certain of the gaseous constituents of the flame.

Preservation of Filtering Cloths.—According to Schraube, this is effected by dyeing with indigo, and a patent has been taken out with that purpose (German Patent, No. 14,843, 1881).

To Separate Silver from Lead.—Donath recommends to add to the solution 4 c.c. of pure glycerine, then an excess of ammonia, and 10 to 15 c.c. of concentrated sodic or potassic hydrate solution, and boil for three to five minutes. Cool and filter and wash the silver thus completely separated, first with hot water, then with acetic acid, and again with water. The lead may be separated from the filtrate by H_2S. The process is also applicable when copper and bismuth are present.—*Monatschrifte für Chemie*, 1880, p. 789.

For the Electrolytic Estimation of Zinc, Reinhardt and Ihle recommend the use of a solution prepared by adding to the neutral zinc chloride or sulphate, sufficient potassium oxalate to dissolve the precipitate which first forms. Free sulphuric acid and ammonium acetate interfere with the deposition. It is recommended that the negative electrode (of platinum) should be covered with a film of copper.—*Jour. Pr. Chem.* [2], xxiv., p. 193.

Nitric Acid in nitrates of soda, potash, etc., can be estimated by dissolving about 0.5 grammes of the substance in 80 c.c. water, adding 10 c.c. sulphuric acid (1 to 3 by volume) and a strip of zinc, and allowing the solution to stand over night. A test with a few drops of solution of permanganate will then show whether any nitrous acid remains. If it does, a little more acid must be added, and the action continued until the reduction to ammonia is complete. The solution may then be rendered alkaline with caustic soda or potash, and the ammonia distilled into a U-tube containing 10 cc. half normal sulphuric acid, which may be titrated by half-normal alkali solution.—J. West Knights in *Analyst*, vi., p. 217.

To Generate Sulphuretted Hydrogen by the use of members of the paraffin series, Lidow recommends fusing the sulphur in a flask, and then allowing the oil to fall upon it drop by drop, about three to five drops per minute. A satisfactory method of managing this process, otherwise so troublesome, and in inexperienced hands even dangerous, is said to be thus secured.—*Journal of Russian Chemical Society*, 1881 [1], p. 514.

Destruction of Microscopic Organisms in potable water can be affected, according to Langfeldt, by the addition of 0.5 gramme of citric acid per litre.—*Chem. Centr.*, 1881, p. 78.

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To Test Flours Microscopically, it is proposed by Steenbuch (*Ber. Deuts. Chem. Gesell.*, 1881, p. 449), to boil up a sample of the flour with water, then to cool and add extract of malt, which will convert the starch into dextrin and maltose, and, after the lapse of a few minutes, to decant, treat with a one per cent. solution of soda to remove the larger proportion of the albuminoid bodies, and then to examine the sediment under the microscope. Many foreign substances and adulterants are thus concentrated, and left in a form suitable for identification.

Spontaneous Combustion by Nitric Acid.—Experiments made by Professor Haas, of Carlsruhe, show that hay, tow, paper-scraps, and similar materials often used for packing, when saturated with nitric acid, and packed tightly, will spontaneously take fire. In some cases where strong acid was used, flames appeared in about twenty minutes, in some other experiments even a shorter time sufficed to produce flame, or a glow which was easily fanned to flame.—*Analyst*, vi., p. 231.

Magnesia may be economically prepared according to a process recently proposed in the *Chemical News* (xliv., p. 76), by treating magnesium chloride solution with calcined dolomite. The interchange of position between the calcium and magnesium affords, it is said, a very pure article.

Tanning.—A very efficient extract, more concentrated than it is otherwise possible to obtain, may be prepared, according to Sadlow, by extracting with water containing 0.5 per cent. of borax.—*Dingl. Polyt. Journ.*, ccxl., p. 228. E. W.

Electricity Applied to Analysis.—Dr. A. Classen, of Aix-la-Chapelle, has just published a pamphlet on this subject ("Quantitative Analyse auf Electrolytischem Wege," 8vo, 52 pp., Aachen, 1882). The work is arranged so as to show the methods of procedure in examining various alloys, as coins, bronze, brass, etc., as well as of iron ores, and other compounds of that character. From the work we abstract the following:

Lead is deposited from solutions containing nitric acid in the form of PbO_2 on the positive pole. If much of the metal is present, the oxide will not adhere, and the method is therefore only applicable when small quantities are present. From the solution of the double oxalate (lead and ammonium or potassium) the metal may be deposited, but in such a form that it is impossible to prevent oxidation and consequent error. Where much lead is present, the ordinary method of precipitation as sulphate is therefore recommended.

Silver can be deposited from the ammoniacal KCy solution. A weak current must be used to obtain an adherent deposit.

Mercury can be deposited from a solution weakly acid with HCl. The deposit must be washed without interrupting the current, and the water completely removed by repeated washing with alcohol and ether.

Tin may be separated from a solution acid with HCl, but the acid must be entirely washed off before the current is broken, or the metal will redissolve. It may also be deposited from the double oxalate (with ammonium). The corresponding potassium-salt cannot be

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used, since it affords a basic salt at the positive pole which dissolves with great difficulty.

Antimony may be deposited from the solution of its chloride to which an excess of potassium oxalate has been added. The addition of an alkaline tartrate gives a deposit adhering more firmly to the platinum dish, but the deposition is much slower. The metal may also be deposited from a solution of sulphantimoniate by a tolerably weak current; arsenic, if present, must first be removed.

Copper is deposited from acid solution of the sulphate which may contain some nitric acid. When the latter acid is present, if zinc also exists in the solution, that metal will be deposited as soon as all the copper is down. Copper may also be deposited from a solution of the double oxalate (potassium or ammonium), to which 3 or 4 gms. crystallized ammonium oxalate has been added. A strong current is required in this case.

Bismuth may be separated from the solution of sulphate or chloride, to which an excess of ammonium oxalate has been added. A large surface for the deposition of the metal is necessary, and its tendency to oxidize is so great that it requires the most careful washing with alcohol and ether to remove all the water.

Cadmium may be deposited from a neutral solution to which an excess of ammonium oxalate has been added.

Iron may be separated from its solution in HCl or H_2SO_4 by removing or neutralizing the most of the free acid, adding sufficient potassium or ammonium oxalate to form the double oxalate, and then 3 or 4 gms. of ammonium oxalate, and electrolyzing hot. Either ferrous or ferric solutions may be used. The former gives a red solution, the latter a green. By the action of the battery, the green solution passes through red to colorless, which indicates that all the iron is deposited. One or two Bunsen cells will deposit several grammes of iron in one or two hours. Manganese, if present, will not begin to separate (as MnO_2) at the positive pole until all the iron is deposited.

Manganese may be deposited from solution of the double oxalate as MnO_2 on the positive pole by a strong current. As the deposit is somewhat pulverulent there is often danger of mechanical loss, and it is recommended in many of the separations not to attempt to precipitate all the manganese by the battery, but to take the solution from which some other metal has been deposited (Fe, Zn, etc.), and to separate the Mn from that by some other analytical method, adding to it of course what MnO_2 has been deposited on the positive pole of the battery. Aluminum phosphate interferes decidedly with the correct estimation of manganese by the battery.

Alumina separates from an oxalate solution, apparently in consequence of a decomposition of the oxalate which holds it in solution.

The electrolysis of ammonium oxalate solution produces ammonium bicarbonate, and so long as the ammonium oxalate remains in excess, no alumina precipitates. A very large excess of ammonium oxalate is therefore required to prevent the alumina from separating in electrolyzing alkaline or neutral solutions.

Chromium is oxidized in oxalate solution by the action of the battery to chromate, in which form it may be separated from many of the elements or compounds with which it occurs. The presence of nitrates must be avoided.

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Nickel and Cobalt may be separated by the battery from ammoniacal solution of the double sulphate or from the neutral or alkaline solution of the double oxalate (ammonium or potassium) containing considerable excess of ammonium oxalate. In the latter case the solution should be kept hot.

Zinc is separated by essentially the same method. The solution of sulphate is neutralized by KHO, enough ammonium oxalate added to form the double oxalate, and then 3 to 4 gms. more solid ammonium oxalate, and electrolyze hot.

The reaction taking place in the solution of double oxalates is described in such a way as to lead us to infer that it may be thus represented—*e.g.*, in the case of zinc:

$$ZnC_2O_4.(NH_4)_2C_2O_4+2H_2O=Zn+2NH_4HCO_3+2CO_2+H_2.$$

We are glad to see that to Professor Gibbs is justly accredited the first application of electrical separations to analysis. The battery is undoubtedly destined to play a more important part in quantitative analysis than it has yet done, and any information such as the pamphlet before us is most welcome. E. W.

METALLURGY.

Scaffolds and Engorgements in Blast Furnaces.—A number of interesting papers on this subject have been read during the last year before the Institute of Mining Engineers. Mr. Witherow, of Pittsburg, Pa., sums up the different methods used, which summary we condense as follows, adding details from the other papers.

First method.—Breaking into furnace lining 15 to 18 ft. above tuyeres, inserting a dry tuyere or gas-pipe, and injecting all the air possible into the furnace. If the tuyere thus inserted is below the scaffold, the blast will soon cut or melt its way through it; if above the obstruction, the rapid smelting and liquefaction will soon work its way downward. In both cases the result is to detach the agglomerated mass from the brick masonry, and under the weight of the charge above the scaffold generally gives way.

Second method.—Blowing down. Charging is suspended and the material in furnace allowed to sink until the surface of the charge has reached the lower section of the zone of fusion. All material adhering to walls is undermined and cut away by the intense heat developed.

Third method.—Use of high explosives, giant powder, etc., in the hearth, as practised by Mr. T. F. Witherbee, at the Crown Point Furnace. The explosive is enclosed in wooden cases, made by boring a hole in a pine plug of wood and shaving down the outside till one-quarter to one-eighth inch thick. For a larger charge of twelve pounds a box of one-half inch pine was used. Generally the fuses were fired before the cartridges were put in the holes, and a shovelful of sand thrown against the hole formed the tamping. In one case regular mining work had to be undertaken, a narrow opening being driven through the red-hot pasty mass; no less than twenty-six shots of one-quarter to one-half pound of giant powder being required. In this case the last shot broke through and tapped out molten iron.

Mr. Witherbee describes the firing of the big shot as follows:

The salamander was found to rest on a bed of anthracite coal

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about even with the bottom of the cinder-arch. One of the helpers was set to digging a hole just under the salamander, which he succeeded in doing without much difficulty by working out the lumps of coal clear to the centre of the crucible and dipping down about 34 inches below the tuyeres. Into this a charge of 12 pounds, equal to 22 cartridges, of "giant" and "Monaky" powder, went at 2 P.M.

This cartridge was made by removing the paper from the common cartridge and ramming the loose powder into a box 5 x 6 x 27 inches, made of ½-inch pine, to which was fastened an iron handle. Two exploders were put in it, with two lines of fuse, the outer ends being wound close together and cut off square, so that both could be fired simultaneously. The hole was very hot and contained a few quarts of melted iron at the bottom. Cooling off with water would not do, as the "Monaky" powder would explode in boiling water, so a trial was made with wet clay, with which the hole was filled, and then a hard-wood scantling 6 x 8 inches was driven in and removed to make room for the box. The clay could not be made to stick on to the top of the hole, and would not entirely cover the iron in the bottom, and as matters were getting worse all the time it was decided to make the attempt. By means of the handle the box was thrust to the bottom of the hole and the fuses fired by a hot iron, while an assistant, Mr. Duffy, with a hoe, packed sand in the mouth of the hole. Probably less than ten seconds was used in loading, tamping, firing, and fleeing from the wrath to come.

Some had predicted, as the result, damage ranging from total destruction of the stack down to a mere fizzle. So, for want of better occupation, the upper part of the stack was watched with considerable interest. The explosion was not very loud, and no movement of the stack visible to the eye took place, although the ground shook considerably, seemingly before the noise was heard. The wind was on the furnace at the time, but probably little or none going in, and upon getting back to the stack it was plain that things had changed. Gas was coming out around every tuyere and through all the joints of the casing, from the tuyere breasts down, and for one-half the way around the stack mushy cinder, lime, and an occasional rounded lump of small coal laid from one to two feet thick, the total quantity being about fifty cartloads. The stock had gone down an unknown distance, and the salamander about as far up, and good coal appeared in front of No. 6 and No. 2 tuyeres for the first time in forty-eight hours. The damage was confined to the breaking off of two ¾-inch water-pipes and of one lug of No. 6 tuyere pipe. The furnace was again filled, and has worked very smoothly up to the present time.

As the result of his experience in the use of this method, Mr. Witherber recommends as the best treatment in all similar cases where the crucible becomes solidly filled with a gummy mass, entirely shutting out the blast:

1. To try the effect of a heavy explosion near enough to the centre of the furnace to make it possible to break a hole up through, or at least to disrupt the mass enough to permit the blast to pass through, which will soon melt it out. The reason I recommend this plan first is that its success depends almost wholly upon getting a hole properly located, and of course the sooner this is begun the more chances for

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accomplishing it. To be sure, raising the tuyeres will, once in awhile, do as well, but that is often difficult, and sometimes impossible, owing to the construction of the furnace.

2. Should the first fail, or should it be impossible to get a hole in the proper location early enough, then there remains the slower, though probably surer, method of excavating upward by blasting through to good fuel. This opinion is based upon the fact that such obstructions are seldom of any great thickness. In loading holes overhead it is found sufficient to simply prop the cartridge box in the hole, usually by a ¼-inch iron rod serving as the handle, and dispense entirely with tamping. Small drills, say 1¼-inch, work the best, and in case a large hole is required it is best to drill two or three close together and break out the core by firing light charges in one.

As to the danger to life or property, it may be dismissed as of no account, providing that the precautions indicated are always taken, and the loading done by a reasonably cool-headed person.

Fourth method.—Removal of tuyeres, opening of front, and shovelling out of materials so as to allow the scaffold to fall. This plan generally succeeds when a furnace bridges during the blowing in, but when a scaffold has fairly established itself, and especially when the stock has become agglomerated, the work becomes difficult. Such a case happened at the Lucy Furnace near Pittsburg some years ago, and was described by Mr. Pechin,[1] from whose paper we quote the following:

"The furnace had been working well on low-grade ores of about 50 per cent., producing daily 68 to 70 tons. The charge was changed to Republic ore, one of the purest and best of the Lake Superior ores, for the purpose of making Bessemer iron. This was charged by itself, and Mr. Skelding, the founder, reports that he did not succeed in getting a single cast when it came down before the furnace chilled from the hearth to the top of the boshes, some 25 feet. Every effort was made to overcome the difficulty without avail, and the disagreeable work of cleaning out the furnace was begun. The hearth was dug out some six or eight feet up, when Mr. Skelding remarked in the hearing of one of the proprietors that he wished he had a cannon. A mortar was forthwith procured from the arsenal, and they commenced firing shots up into the chilled mass. But by and by the mass became pasty, and the cannon-balls, of which they had but three, stuck fast. Mr. Skelding put in a large charge of powder, and then, to the astonishment and amusement of the spectators, rammed the mortar full of cotton waste, and on top of this put a lump of hard ore weighing about fifty pounds. It is not certain whether he said 'presto, change,' but this novel shot brought down the scaffold and the cannon-balls, and Lucy once more breathed freely."

[1] Transactions Am. Inst. M. E., II., p. 59.

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SCHOOL OF MINES NOTES.

THE old School of Mines building has been demolished, and we have cleared our eyes of the dust raised by the energetic Irishmen enough to see that the excavation for the new building has already made some progress. The plans for the new building, however, are not as yet ready for publication, so that we shall have to postpone till next year our promised description and illustrations.

THE College library building has made great progress since last mentioned in these pages, the walls being nearly complete, with every prospect of its being ready for use next year, when it will be occupied temporarily by the Law School until the new building for that department shall be ready.

ALTHOUGH it often happens that students, in the course of their laboratory work, are engaged in operations of more or less danger, serious accidents have fortunately been of rare occurrence, but one having happened this year. This took place on April 3d, and consisted in the explosion of a glass retort in which Mr. W. H. Cooper, of the graduating class, was distilling a mixture containing, among other ingredients, a considerable proportion of concentrated sulphuric acid. Mr. Cooper's face and arms were covered with the boiling liquid, and were somewhat cut by flying glass. But for the prompt and careful treatment he received at the hands of Mr. Elliott, and others, his injuries, so his physician states, would have been very serious, and might even have resulted fatally. The burns were first washed carefully with water, and then with a solution of sodium bicarbonate to neutralize the acid, after which an emulsion of lime-water and oil was applied.

The efficiency of this method of treatment is shown by the rapidity with which the burns are healing, and it is to be strongly recommended in all similar cases.

It is unnecessary to assure Mr. Cooper of the hearty sympathy of his classmates and all others at the School who know him, and their hope that he will very soon be able to resume his work in their midst.

THE CHEMICAL SOCIETY held its Sixty-eighth Regular Meeting on January 26, 1882. President Wittmack in the chair. Mr. McKenna, '83, read a paper on the "Use of Woody Fibre in Paper-Making," illustrated with specimens, etc. Mr. Cooper announced his paper on "Dry Assay of Copper."

Several members were elected, and a lecture committee, consisting of Messrs. Powers, Painter and Wittmack, was appointed.

The Sixty-ninth Regular Meeting was held on February 16, 1882. Several new members were admitted, and the following officers were elected for the present term:

President, C. V. V. POWERS, '82.
First Vice-President, F. SANDS, '82.
Second Vice-President, H. FEUCHTWANGER, '82.
Secretary, F. W. TRAPHAGEN, '82.
Treasurer, G. E. PAINTER, '83.

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The Seventieth Regular Meeting was held on March 9, 1882. A number of active and honorary members were elected and the Society then adjourned.

The Society held its Seventy-first Regular Meeting on April 6 1882. Having finished the routine business, the lecture committee recommended that the following gentlemen be asked to lecture on the subjects:

PRESIDENT F. A. P. BARNARD, "Germ Theory in Disease;" DR. DRAPER, "Solar Spectrum Photography;" DR. STEBBINS, "Azo Colors;" DR. LOVE, "Gas Analysis." Adjourned.

The Seventy-second Regular Meeting was held on April 20, 1882. After the regular business, Mr. Bullman read his paper on the "Atomic Theory." Mr. Calman announced his paper on "Cyanine," as also Mr. Wittmack on "Coal-tar—its Influence on the Industries of the World." Then followed adjournment.

ADDITIONS TO LIBRARY SINCE FEBRUARY.

ENGINEERING.

Abbot, Lieut.-Col. Henry L. Report upon a System of Submarine Mines for Defending the Harbors of the United States. 4to. Washington, 1881.

Dannenberg, Julius. Bergmännisches Wörterbuch. 12mo, 464 pp. Leipzig, 1882.

Letters as to the Practicability of a Ship Railway. 8vo, 46 pp. St. Louis, 1882.

Porter, Charles T. A Treatise on the Richards Steam Engine Indicator. 8vo, 258 pp. London.

Robert, Major Henry M. Analytical and Topical Index to the Reports of the U. S. Chief of Engineers, 1866–79. 8vo, 624 pp. Washington, 1881.

Shock, W. H. Steam Boilers, their Design, Construction, and Management. 4to, 473 pp. New York, 1881.

Varona, Adolfo de. Sewer Gases. 18mo, 144 pp. New York, 1882.

Woodward, C. M. A History of the St. Louis Bridge. 4to, 391 pp. St. Louis, 1881.

GEOLOGY AND NATURAL HISTORY.

Bessey, Charles E. Botany for High Schools and Colleges. 8vo, 611 pp. New York, 1881.

Bulletin of the Torrey Botanical Club. Vols. 1–6. 8vo. New York. 1870–81.

Darwin, Charles. The Formation of Vegetable Mould. 8vo, 326 pp. New York, 1882.

Fisher, Rev. Osmond. Physics of the Earth's Crust. 8vo, 299 pp. London, 1881.

Heddle, M. Forster. Geological Map of Sutherland. 1881.

Judd, John W. Volcanoes: What they Are and What they Teach. 8vo, 381 pp. New York, 1881.

Sandberger, Fridolin. Untersuchungen über Erzgänge. Erstes Heft. 8vo, 158 pp. Wiesbaden, 1882.

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Liebisch, Dr. Th. Geometrische Krystallographie. 8vo, 464 pp. Leipzig, 1881.

Rammelsberg, C. F. Handbuch der Krystallographische-Physikalischen Chemie. 8vo, 615 pp. Leipzig, 1881.

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Grant-Francis, George. The Smelting of Copper in the Swansea District of New South Wales, from the time of Queen Elizabeth to the Present Day. 8vo. London, 1881.

Neveu, F., and Leon Henry. Traité Pratique du Laminage du Fer. 8vo and Atlas. Paris, 1881.

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Allen, Alfred H. An Introduction to the Practice of Commercial Organic Analysis. 2 vols. 8vo. London, 1879.

Barfoed, Chr. Th. Lehrbuch der Organischen qualitativen Analyse. 8vo. Kopenhagen, 1881.

Barreswil et Aimé Gerard. Dictionnaire de Chimie Industrielle and Introduction. 5 vols. 8vo. Paris, 1861.

Beilstein, Dr. F. Handbuch der Organischen Chemie. Lief. 1–9. Leipzig, 1881.

Bierdermann, Dr. Rudolph. Chemiker Kalender. 2 vols. 18mo. Berlin, 1882.

Blyth, Alexander Wynter. Analysis of Food and Detection of Poisons. 8vo, 468 pp. London, 1879.

Binns, William. An Elementary Treatise in Orthographic Projection. 2 vols. 8vo. London, 1876–78.

Butlerow, A. Lehrbuch der Organischen Chemie. 8vo, 752 pp. Leipzig, 1868.

Dumas, M. Traité de Chimie Appliquée aux Arts. 8 vols. 8vo. Paris, 1828.

Fittig, Rudolph. Wöhler's Outlines of Organic Chemistry. Translated by Ira Remsen. 8vo, 522 pp. Philadelphia, 1873.

Girardin, M. Y. Leçons de Chimie Élémentaire appliquée aux arts Industriels. 5 vols. and Supplement. Paris, 1880.

Herzberg, Dr. Carl. Vollständiges Handbuch der Chemischen Fabrikenkunde. 8vo, 879 pp. Weimar, 1858.

Hétét, Frederick. Manuel de Chimie Organique Élémentaire avec ses Applications à la Médecine, à L'Hygiene et à la Toxicologie. 8vo, 767 pp. Paris, 1880.

Hoffman-Schaedler Tabellen für Chemiker. 8vo, 312 pp. Berlin, 1877.

Kekulé, Dr. Aug. Chemie der Benzolderivate. 2 vols. 8vo. Stuttgart, 1867–1881.

Lormé, G. Eugène. Manuel Complet du Fabricant de Produits Chimique. 4 vols. 18mo, and Atlas. Paris, 1861.

Oppolzer, Theodore. Lehrbuch zur Bahnbestimmung der Kometen und Planeten. 4to. Leipzig, 1870–1880.

Remsen, Ira. Principles of Theoretical Chemistry, with special reference to the Constitution of Chemical Compounds. 8vo, 231 pp. Philadelphia, 1877.

Richter, Dr. V. v. Chemie der Kohlenstoffverbindungen. 8vo, 834 pp. Bonn, 1880.

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GRADUATE DEPARTMENT.

LACK of space again compels us to omit extracts from graduates' correspondence, except so far as containing matter of professional interest, and printed among the original papers or in the Professional Notes.

WE have received the third annual report of the Permanent Secretary of the Class of 1878, issued in March, 1882. In addition to the usual letters from members of the class, the report contains a history of the Class Society of 1878, from October, 1874, to June, 1875, by B. Willis, the "Franco-German" petition sent to the Faculty by the class, and an account of the experiences of the M. E.'s at Drifton in 1877, by "Commodore" Perry. In the "Postscript in lieu of a Preface," the Permanent Secretary, Mr. Munsell, asks the opinion of the class as to whether the report shall be published separately as heretofore, or shall in future be incorporated into the QUARTERLY.

The pamphlet is well arranged and carefully edited, and reflects credit upon its compiler.

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1869.

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1878.

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1879.

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1880.

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1881.

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Herbert M. Wilson, C.E., Sinalo and Durango Railroad, Culiacan, Mexico.

ENUMERATION.

Class of 1867	13	Class of 1876	26
" 1868	20	" 1877	31
" 1869	11	" 1878	40
" 1870	9	" 1879	33
" 1871	7	" 1880	36
" 1872	5	" 1881	36
" 1873	5		
" 1874	9	Total number of graduates	301
" 1875	20		

Known to be deceased.

Class of 1868	Archibald MacMartin.
" 1869	Thomas Monahan Blossom.
" 1869	Henry Newton.
" 1872	Frank B. Jenney.
" 1875	Robert S. Lamson.
" 1877	Henry G. Clark.
" 1877	James R. Priest.
" 1878	Owen F. Olmsted.
" 1879	Francis B. Stewart.
Total	9

Number of graduates living 292

Employed as follows:

As Mining Engineers, consulting	22	
" " " field-work	10	
" " " superintendents of mines	17	
" " " instruction	4	53
" Civil and Mechanical Engineers, consulting	2	
" " " " field-work	32	
" " " " instruction	7	
" " " " sanitary work	4	45
" Chemists, consulting and analytical	38	
" " superintendents of works	3	
" Assayers	32	
" Chemists and Assayers, instruction	15	88
" Metallurgists, consulting	7	
" " superintendents of works	10	17
" Geologists and Mineralogists, consulting	2	
" " " " field-work	4	
" " " " instruction	6	12
" Architects	4	
" Lawyers	5	
" Artists	1	

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As Physicians	1
" Editors	1
" Ministers	1
In Agriculture	2
" Business	30
" Manufacturing	8
" Studying, Europe and elsewhere	15
" Travelling	2
Occupation unknown	7
Total	292

	Per cent.	
Employed as Mining Engineers	17.67	
" " Civil and Mechanical Engineers	15.00	
" " Chemists and Assayers	29.34	
" " Metallurgists	5.67	
" " Geologists and Mineralogists	3.99	
" " Architects, Lawyers, etc	4.99	
" in Professional work, Total		76.66
" " Business and Manufacturing		12.34
" " Studying and Travelling		5.67
Occupations unknown		2.34
Deceased		2.99
		100.00

Degrees conferred to date.

E.M	198
C.E	79
Ph.B	63
Ph.D	27
Total	367
Deduct for repetitions	66
	301

In the classification of the graduates as to employment, the principal occupation only of each man is taken into consideration. Many of the mining engineers act as chemists, geologists, etc., and vice versa. The list is correct to April 15, 1882.

P. DE P. RICKETTS,
Resident Secretary Alumni Association.

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GRADUATES.—ALPHABETICAL LIST.

ADAMS, J. M., '67
Adams, W. J., '78
Allen, '74
Andresen, '81
Aschman, '81
Austen, '72
Austin, '76

BARNARD, '68
Barros, '77
Baxter, '68
Beard, '77
Beebe, '80
Behr, '77
Benedict, '74
Benjamin, M., '78
Benjamin, F. P., '80
Bleecker, '81
Blossom, '69
Blydenburgh, '78
Booraem, '78
Braschi, '81
Bridgham, '67
Brinckerhoff, '78
Britton, '79
Brown, '67
Browning, '80
Brownson, '67
Bruckman, '69
Bruen, '76
Brugman, '80
Buckley, '77
Bush, '81
Butler, W. P., '78
Butler, N., '80

CADY, '77
Cameron, '74
Campbell, '69
Canfield, A. C., '77
Canfield, F. A., '73
Carson, '68
Cauldwell, '77
Chazal, '81
Chester, '68
Church, '67
Churchill, '80
Clark, '77
Clarke, '80
Cloud, '79
Colby, C. E., '77
Colby, A. L., '81
Colton, '73
Constant, '77
Cornell, '77
Cornwall, G. R., '76
Cornwall, H. B., '67
Cornwall, H. C., '79
Coursen, '68
Craven, '76
Cushman, '78

DAVIS, '78
De Luze, '79
Devereux, '78
Douglas, '81
Downing, '78
Drummond, '78
Dunham, '81

EASTWICK, '79
Elliott, W. G., '78
Elliott, A. H., '81
Engel, '80

FALES, '71
Fernekes, '78
Floyd, '77
Foote, '76
Francke, '80
Furman, '81

GARLICHS, '80
Garrison, '76
Geer, '68
Giddings, '67
Goldschmidt, '71
Gordon, '71
Gracie, '67
Gratacap, '76
Greene, '80
Greenleaf, '80
Griswold, '81

HAAS, '78
Haffen, '79
Haight, '70
Hale, '67
Hall, '76
Hallock, '80
Hamilton, '76
Hanna, '68
Harker, '79
Harmer, '67
Hasegawa, '78
Hathaway, '79
Heath, '67
Helleberg, '77
Hemmer, '81
Hendricks, '80
Hildreth, '77
Hodges, '78
Holbrook, '76
Holden, '78
Hollerith, '79
Hollick, '79
Hollis, '78
Hooker, '69
Hooper, '80
Hopke, '80
Hoyt, '76
Hudson, '80
Hunt, '76
Hutton, '76

IHLSING, M. C., '75
Ihlsing, A. O., '77
Iles, '75
Ingersoll, '70
Irving, '69

JACKSON, '75
Jenney, W. P., '69
Jenney, F. B., '72
Johnson, E. M., '78
Johnson, G. H., '78
Johnson, I. B., '79
Johnston, '79
Jordao, '77
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KARR, '78
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King, '76
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Meissner, '80
Merwin, '79
Meserole, '81
Milliken, '79
Moffatt, '68
Morewood, H. F., '76
Morewood, G. B., '78
Morris, '78
Mott, '73
Munroe, H. S., '69
Munroe, O. M., '79
Munsell, '78
Murphy, J. G., '77
Murphy, H. M., '78
Murray, '74

NAMBU, '78
Navarro, '80
Neftel, '79
Nesmith, '79
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Newberry, '78
Newton, '69
Neyman, '81
Nichols, '77
Noble, '79
Norris, '77
Noyes, W. S., '75
Noyes, J. A., '78

O'CONNOR, M. J., '81
O'Connor, T. D., '81
O'Grady, '76
Olcott, '74
Olmstead, '78
Owen, '78

PALMER, '78
Parker, R. A., '78
Parker, A. McL., '80
Parks, '80
Parrot, '70
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Pazos, '78
Perry, '78
Pfister, '75
Pistor, '68
Pitkin, '81
Platt, '68
Pool, '75
Potter, '69
Potts, '80
Priest, '77
Putnam, '75

RADFORD, '77
Randolph, J. C. F., '69
Randolph, J. Fitz, '76
Raymer, '81
Reed, S. A., '77
Reed, W. B. S., '79
Rees, B. F., '74
Rees, J. K., '75
Rhodes, F. B., '74
Rhodes, R. D., '79
Richmond, '81
Ricketts, '71
Riggs, '71
Roberts, G. S., '71
Roberts, A. C., '81
Robertson, K., '68
Robertson, R. S., '71
Robinson, '80
Rogers, '77
Rolker, '75
Ross, '76
Russell, '75
Rutherford, '79
Ruttman, '80

SAGE, '77
Sawyer, '81
Schermerhorn, '68
Schneider, '76
Shack, '68
Share, '81
Sheldon, '79
Singer, G. H., '80
Singer, G., '80
Sloane, '72
Smalley, '80
Smeaton, '77
Smith, W. A., '68
Smith, L., '68
Smith, M., '80
Smythe, '77
Stallknecht, '68
Starr, C. D., '81
Starr, H. F., '79
Stearns, '81
Stewart, '75
Stone, '79
Strieby, '78
Suydam, '79
Swain, '81

TERHUNE, '70
Thacher, '77
Thompson, '75
Tilden, '76
Tonnelé, '80
Torrey, '80
Tucker, '75
Tuttle, W. W., '67
Tuttle, E. G., '81

VAN ARSDALE, '68
Van Blarcom, '76
Van Boskerck, '77
Van Lennep, '67
Van Sinderen, '81
Van Wagenen, '70
Vulté, '81

WALBRIDGE, '76
Walker, '80
Waller, '70
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Waterbury, '77
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Wheeler, M. D., '68
Wheeler, H. A., '80
Wiechmann, '81
Williams, J. T., '73
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Williams, G. W., '79
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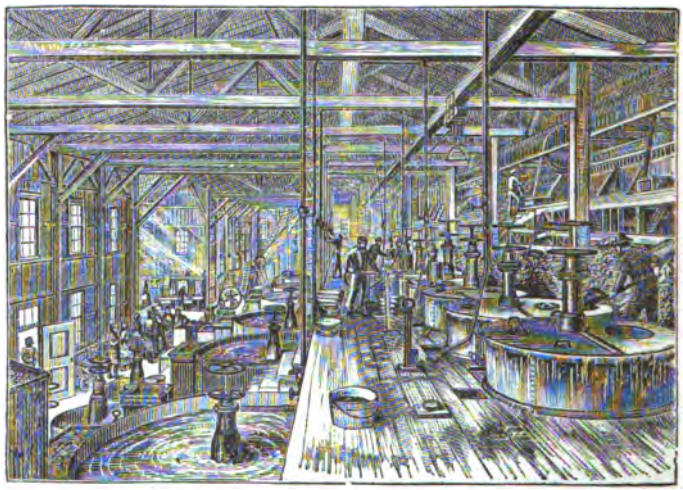

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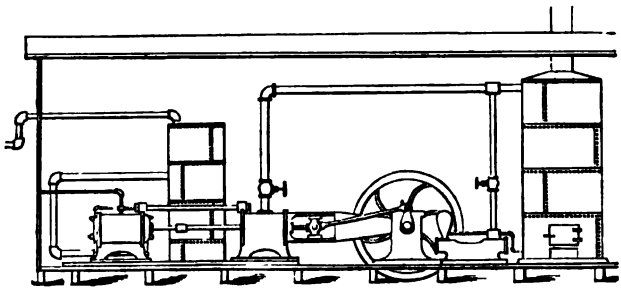

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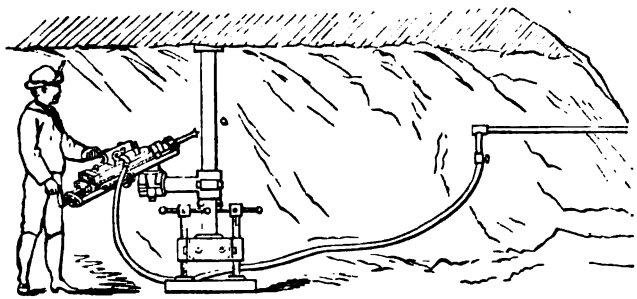

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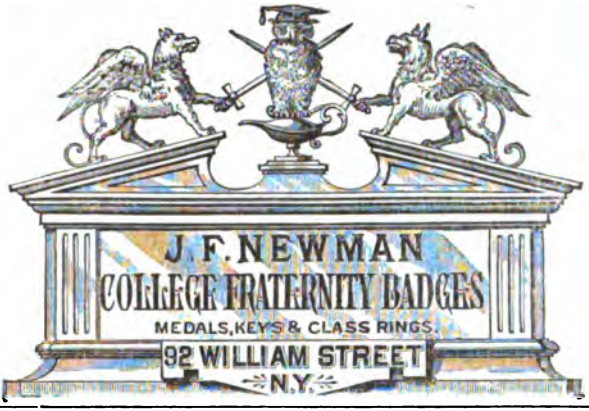

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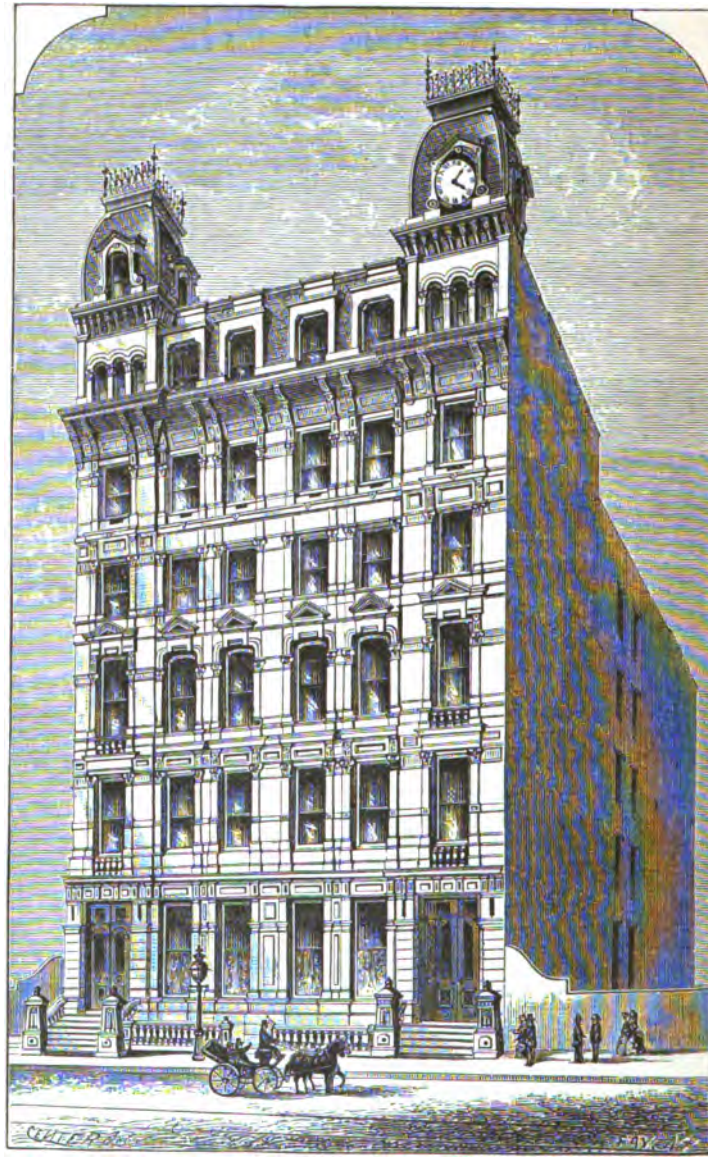

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